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Final Report
TASK 1A - VBM Design Guidelines
SUMMARY

by

Inco Limited, Thompson Division
Thompson, Manitoba

March 1990

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RESEARCH & DEVELOPMENT IN MANITOBA MINES

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VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

SUMMARY

This chapter has been prepared as part of the

TASK 1
To Develop Vertical Block Mining Design Guidelines
For A Steeply Dipping Orebody;
And To Develop Scaling Techniques
of the

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Prepared by

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VERTICAL BLOCK MINING (VBM) DESIGN GUIDELINES

The Vertical Block Mining (VBM) Design Guidelines consists seven chapters, published separately. The seven chapters are:

Summary

1. **Cut and Fill Mining**
2. **Data Collection Systems**
3. **VBM Stope Design**
4. **Drilling and Blasting**
5. **Support Requirements**
6. **VBM Stoping Procedures**

Reference to this chapter should be quoted as follows: "**Vertical Block Mining Design Guidelines Summary**"; Canada/Manitoba Mineral Development Sector B - Mining; DSS File No. 14SQ.23440-4-9147-3; CANMET Project No. 4-9147-3; March, 1990

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ABSTRACT

A study into the application of Vertical Block Mining at Inco's Thompson Mine has been undertaken with the objective of developing design guidelines. The intent of these guidelines is to provide information that could be used by Thompson to increase efficiencies or, by another mine, to examine in detail the application of the method to steeply dipping orebodies.

The study addresses six specific topics. These are:

- Cut-and-fill mining
- Data collection systems
- VBM stope design
- Drilling and Blasting
- Support Requirements
- Stoping Procedures

Brief descriptions of each topic are given below:

Cut-and-Fill Mining

The objective of this section of the study is to describe cut-and-fill stoping as it is practiced in Manitoba and to examine the potential for improvements, especially of the procedure used at Thompson in those areas where VBM stoping is not appropriate. Cost and productivities applicable to Manitoba cut-and-fill operations are developed and compared with experience elsewhere.

Two cut-and-fill operations are described: the Thompson Mine and the Trout Lake Mine. These mines represent the application of the cut-and-fill method to narrow and wide orebodies respectively. The Thompson Mine typifies a mature, semi-mechanized operation with limited access to the working place. The Trout Lake Mine typifies an operation which has been developed by ramp access from surface and uses mechanized equipment for all aspects of the operation.

It was found that the two Manitoba cut-and-fill operations demonstrate a satisfactory level of cost and productivity within the constraints of the methods. It is evident that much higher levels of productivity could be achieved by providing ramp access for the method at Thompson and by increasing mechanization, but it is uncertain whether a decrease in cost could be achieved.

The study clearly demonstrates that cut-and-fill remains a viable method, comparable in achievement with VBM in certain geometrical circumstances.

Data Collection Systems

The development and maintenance of a well structured data collection system is of fundamental importance to a mining operation, regardless of the mining method employed. At Thompson Mine, a data collection system was developed based on the case histories from the mining of 126 vertical crater retreat blocks.

The database was separated into two sections: geotechnical and mine operational data. The geotechnical database included categorizations such as geology, structure, strength and rock mass quality. The concept of stress and the effect of the redistribution of stresses through the use of visual monitoring and field instrumentation techniques was included in this geotechnical database. The mine operational database was separated into categories such as stope geometry, development, drilling, blasting and mucking. The mining efficiency, measured from the combination of parameters from these groupings, was subsequently derived from the mine operational database.

Data collection serves three purposes. Firstly, data assists in understanding the actual situation so that proper evaluations can be made. Secondly, the proper evaluation of data will form the basis for actions and decisions. Too often decisions are made on judgement, based on opinions and experiences with no records for backing. Thirdly, a database allows current data to be compared with historical data to determine whether or not a process is proceeding as normal, or whether the process has actually changed.

The effective use of a data collection system can be summarized best by the following list of activities:

- comparison of individual mining and geotechnical parameters;
- analysis of operating parameters in order to determine levels of control;
- indication of potential problems in current mining areas;
- development of empirical relationships for the purpose of future mine design.

VBM Stope Design

A major factor in the planning of an underground mine is the choice of suitable excavation dimensions. The economy of size suggests that the larger the excavations, the lower the costs. As larger mining equipment can be used, a reduced work force is required. The excavation size, along with the ore geometry, is used to determine the most suitable mining sequence.

A philosophy of stope design has been developed in which past experience is used to calibrate both empirical and numerical models. In turn, these models are used to assess applicable stope dimensions and to predict ground control events associated with different stoping sequences. This must be done within the framework of ore value, tonnage and prediction requirements.

Drilling and Blasting

Currently, nearly all of the ore in VBM stopes is broken using conventional VCR blasting techniques. Supplemental multi-deck blasting procedures are used to expand blocks laterally by slashing and to recover crowns. It has been shown that cost advantages and higher mining efficiencies (compared with mechanized cut and fill stoping) have resulted from the application of VCR blasting techniques. However, these advantages have been offset in part by very heavy dilution in narrow, sinuous orebodies and reduced ore recovery, compared with traditional mining and/or rock breakage approaches. As noted previously, this is a conscious trade off, but it seems that the original concept of VBM as defined earlier is still appropriate, particularly the observation relating to the application of drilling and blasting practices. It is beyond the scope of this study to detail alternative rock breakage approaches that may be applicable, but it is suggested that full column bench blasting has a future role in stoping operations.

Support Requirements

Tentative guidelines for the selection and design of support in development openings have been developed. These guidelines are based on current industry practice, accepted design procedures and experience gained at Inco's Thompson Mine.

The objective of the support guidelines proposed is to provide a rational basis for selecting support type and density to ameliorate those problems that have the greatest impact on mining, that is back failure, access loss or deterioration, and hangingwall slough. Though primarily oriented towards development excavation support, the guidelines also address the requirements for stope walls.

Any procedure for support design and selection must be largely empirical due to the many variables involved. Five steps can be identified:

1) Characterization of rock mass
2) Prediction of induced stress over life of opening
3) Prediction of likely ground behaviour/failure
4) Support options available
5) Selection

These guidelines therefore provide procedures associated with steps 1 to 4, while the final step, selection, must remain with the user.

Stoping Procedures

Stoping procedures for vertical block mining (VBM) as they are practiced at the Thompson Mine are described. A description of stoping procedures at other similar operations is also provided to illustrate practical mining solutions which have evolved in somewhat different geologic environments. These operations are the Homestake Mine in USA, Carolusberg Mine in South Africa and Cobar Mines in

Australia. The goal of this work is to provide workable solutions to problems which will likely be encountered during the implementation of VBM stoping in a steeply dipping orebody in any hard rock mining environment.

The VBM method has been successfully used at Thompson and at a number of other mines worldwide. A common thread between the mines is the use of VBM to convert from cut-and-fill methods to more productive bulk mining or, to convert areas experiencing significant ground control problems. Interestingly, though several mines were identified that were originally planned as VBM operations (e.g. Rubiales in Spain), few continued with the method beyond initial start-up.

At Thompson, the VBM method has been used in conjunction with crater blasting techniques. Ideally, this approach is suited for wide orebodies (greater than 7.6 m (25 ft.) width) with strong competent footwalls and hangingwalls. Thompson have been successful in applying the method in far from ideal circumstances, that is in narrow orebodies with weak hangingwalls, in order to achieve major productivity gains. The trade-off has, however, been increased dilution. Because of the grade of the Thompson orebody and historical nickel prices, this trade off has generally been acceptable. However, recent high nickel prices have substantially increased the cost of dilution, and the mine has embarked on a program to further investigate the causes and adjustments required to the VBM method in order to minimize the amount of dilution.

SOMMAIRE

Une étude sur l'application de la méthode de minage de blocs par retrait vertical (Vertical Block Mining VBM) à la mine Thompson d'Inco a été entreprise avec pour objectif de développer des procédures de design. Le but de ces procédures est de fournir des informations qui pourraient être utilisées par Thompson ou par une autre mine afin d'augmenter l'efficacité examiner en détail l'application de la méthode pour des gisements à fort pendage.

Cette étude couvre six aspects spécifiques. Ils sont:

- Méthode de minage coupe et remblai
- Système de récolte de données
- Design de chantier VBM
- Forage et sautage
- Soutènement
- VBM, procédure d'abattage

Une brève description de chaque aspect est donnée plus bas.

Méthode de minage coupe et remblai

Cette section a pour objectif de décrire la méthode de minage coupe et remblai telle qu'utilisée au Manitoba et d'étudier les possibilités d'améliorer la méthode utilisée spécifiquement à Thompson. Les coûts et la productivité des opérations de coupe et remblai manitobaines ont été déterminés et comparés avec d'autres opérations.

Les deux opérations de coupe et remblai décrites sont celles des mines Thompson et Trout Lake. Elles présentent respectivement l'application de la méthode coupe et remblai pour des gisements étroits et larges. La mine Thompson présente une opération mature, semi-mécanisée et avec des accès limités au lieu de travail. La mine Trout Lake présente une opération qui a été développée à l'aide d'une rampe d'accès en surface et qui utilise des équipements mécanisés aux différents stades des opérations.

Les deux mines présentent des coûts et une productivité satisfaisantes compte tenus des contraintes de cette méthode. Pour les opérations de la mine Thompson, il est évident qu'un niveau de productivité supérieure pourrait être obtenu par le percement d'un rampe d'accès et par une augmentation de la mécanisation. Cependant il n'est pas évident qu'une réduction des coûts pourrait être obtenue.

L'étude démontre clairement que l'utilisation de la méthode demeure toujours attrayante si les résultats sont comparés avec ceux obtenus à l'aide de la méthode VBM dans certaines circonstances géométriques.

Système de récolte de données

Le développement et la mise à jour d'un système de récolte de données bien structuré ont une importance fondamentale dans une opération minière, peu importe la méthode de minage employée. Pour

la mine Thompson, un système de récolte de données a été développé et appliqué sur 126 blocs minés antérieurement par la méthode des cratères par retrait vertical (VCR).

La base de données a été séparée en deux parties: les données géotechniques et les données opérationnelles de la mine. Les données géotechniques incluent des catégories telles que géologiques, structurales, sur la résistance et la qualité de la masse rocheuse. Les concepts de contraintes et l'effet de la redistribution des contraintes après percement ont été inclus dans la base de données géotechniques à l'aide d'inspections visuelles et des techniques d'instrumentation.

La base de données opérationnelles de la mine a été séparée en catégories tels que la géométrie des chantiers, le développement, le forage, le sautage et le marinage. La mesure d'efficacité du minage, mesurée à partir de la combinaison de paramètres des différents groupes, provient de la base de données opérationelles de la mine.

La récolte de données est utilisée pour trois buts. Premièrement, les données permettent de comprendre la situation actuelle et rendent possible des évaluations adéquates. Deuxièmement, ces évaluations des données vont former une base fiable sur laquelle on peut s'appuyer pour prendre des décisions et des actions. Troisièmement, la base de données récoltées peuvent être comparées avec les données antérieures afin de déterminer si l'opération se déroule normalement ou non ou si la méthode d'opération a subit des changements.

L'utilisation efficace d'un système de récoltes de données peut être résumée par la liste d'activités suivantes:

- la comparaison entre le minage individuel et les paramètres géotechniques.
- l'analyse des paramètres d'opération afin de déterminer le niveau de contrôle.
- l'indication de problèmes potentiels dans les régions de minage actif.
- le développement de relations empiriques pour établir les futurs designs de mine.

Design des chantier VBM

Le principal facteur de la planification d'une mine souterraine est le dimensionnement optimal des chantiers. Le dimensionnement économique suggère que pour de plus grandes excavations, il y aura de plus faibles coûts de production. Dans ce cas, de plus gros équipements miniers peuvent être utilisés, donc la force de travail requise est réduite. La dimension des excavations et la géometrie du gisement sont utilisées pour déterminer la séquence de minage la plus propice.

La philosophie du design des chantiers a été développée en utilisant les expériences précédentes pour calibrer les modèles numériques et empiriques. A tour de rôle, les modèles sont utilisés pour évaluer la dimension des chantiers et pour prédire le contrôle de terrain associé aux différentes séquences de minage. Ceci doit être fait initialement avec le calcul de la valeur du gisement, du tonnage et des prédictions requises.

Forage et sautage

Présentement, presque tout le minerai des chantiers VBM est abattu en utilisant les techniques

conventionnelles de sautage VCR. Des procédures supplémentaires de sautage par bancs multiples sont utilisées pour agrandir les blocs à l'aide d'élargissement latéral et pour la récupération des couronnes. Il a été démontré que l'application des techniques de sautage VCR donnent des coûts de minage avantageux et de hauts rendements, comparés à la méthode coupe et remblai mécanisée. Cependant, dans les gisements étroits et sinueux, ces avantages sont réduits par une très grande dilution si ces résultats sont comparés à ceux obtenus par la méthode coupe et remblai et/ou les approches d'attaque de la roche. Comme il a été noté précédemment, cette perte d'efficacité est connue et acceptée mais il semble que le concept original de la méthode VBM est toujours appropriée particulièrement au point de vue du forage et du sautage. Les limites de ce rapport excluent la description d'alternative pour briser la roche mais il faut remarquer que les sautages par bancs verticaux entiers ont un rôle à jouer dans les développements futurs.

Soutènement

Des étapes de cheminement pour la sélection et le design d'un soutènement pour les excavations souterraines ont été développées. Ces étapes sont basées sur les pratiques courantes de l'industrie, sur les procédures de design admises et sur l'expérience acquise à la mine Thompson d'Inco.

L'objectif des procédures de soutènement est de fournir une base rationnelle pour choisir le type et le nombre de supports requis pour améliorer les problèmes d'instabilité qui ont une grande influence sur le minage. Les instabilités qui peuvent être rencontrées sont les effondrements de toit, les blocs non-soutenus ou la détérioration de l'éponte supérieure. Bien que l'orientation principale était basée sur le développement des techniques de soutènement des excavations, ces procédures sont aussi destinées pour le support des épontes dans les chantiers.

Toutes les procédures pour le design et le choix du soutènement sont grandement empiriques car plusieurs variables sont impliquées. Cinq étapes peuvent être identifiées:

1. Caractérisation du massif rocheux.
2. Prédiction des contraites induites duarnt la vie de l'excavation.
3. Prédiction du comportement et les risques d'effondrement de terrain.
4. Soutènement disponible.
5. Le choix.

Ces étapes de cheminement donnent des procédures pour les étapes 1 à 4. Ces étapes sont répétées jusqu'à ce que le concepteur décide du meilleur soutènement.

VBM, Procédure d'abattage

Les procédures d'abattage pour la méthode de minage par blocs verticaux (VBM) employée à la mine Thompson sont décrites. Une description des procédures d'abattage pour des opérations similaires est aussi présentée afin d'illustrer les solutions pratiques de minage qui sont utilisées dans des contextes géologiques différents. Ces opérations sont la mine Homestake aux États-Unis, la mine O'Kiep en Afrique du Sud et la mine Cobar en Australie. Le but de ce travail est de produire des solutions viables pour les problèmes qui pourraient être rencontrés lors de l'implantation de la méthode d'abattage VBM dans des gisements à fort pendage, dans tous les environnements miniers à roche dure.

La méthode VBM a été utilisée avec succès à Thompson et dans nombreuses autres mines à travers le monde. Un point commun entre les mines est que la méthode VBM a été implantée pour substituer la méthode coupe et remblai pour arriver à une méthode plus productive et accroître la sécurité dans les régions où des problèmes de terrain sont anticipés. Parmi les mines initialement planifiées avec la méthode VBM (e.i. Rubiales en Espagne), il est intéressant de remarquer que seulement quelques unes d'entre elles poursuivent l'abattage à l'aide de cette méthode après le design initial.

À Thompson, la méthode VBM a été utilisée selon les techniques de sautage par cratères. Idéalement, cette approche est adaptée pour des gisements larges (plus grand que 7,6 m (25 pi.)) avec des épontes solides et compétentes. Afin d'obtenir des gains majeurs de productivité, la mine Thompson a implanté cette méthode avec succès tout en ayant des conditions non-idéales, c'est-à-dire un gisement étroit avec une éponte supérieure peu compétente. Ce tranfert de méthode a cependant augmenté la dilution. À cause de la teneur du gisement de Thompson et du prix du nickel de cette période, cette nouvelle méthode a généralement été bien acceptée. Cependant, récemment, les prix élevés du nickel ont substantiellement augmenté les coûts de dilution, et la mine a instauré un programme pour déterminer les causes et les ajustements requis à la méthode VBM afin de minimiser la dilution.

1.0 INTRODUCTION

A major study of the application of Vertical Block Mining (VBM) at Inco's Thompson Mine has been undertaken. This study has been sponsored in part by the Federal government, in part by the provincial government of Manitoba and in part by Inco Limited.

The objective of the study was to develop design guidelines for Vertical Block Mining, based on experience gained at the Thompson Mine. The intent of these guidelines is to provide information that can be used by Thompson to increase efficiencies or, by another mine, to examine in detail the application of the method.

2.0 OBJECTIVES

2.1 General

Inco's Thompson Mine is a primary producer of nickel ore originating from both open pit and underground operations. The mine crown pillar is currently being extracted using open pit mining methods, while the ore zone below the 400 level is being mined using conventional underground stoping techniques. Production from the underground mine averages 6,000 tons per day, with the open pit producing a comparable tonnage.

The Thompson Mine ore zone extends over a strike length of 3.5 miles. It is located in a schist zone conformable with the local formation. The ore pinches and swells both along strike and down dip, with further irregularities due to local tension cracks into the wall rocks. Ore zone widths range from the minimum economic mining width of approximately 1.5 m (5 ft.) up to a maximum of 46 m (150 ft.).

The underground mine is serviced by two shafts, as shown in Figure 1. The T1 shaft, at a depth of 1,350 m (4,427 ft.) is located at the south end of the ore zone and serves as the main production shaft. The T3 shaft, located two miles north of T1, operates as a service shaft for the northern half of the ore zone to a depth of 795 m (2,607 ft.). A ramp is currently being developed at the T3 shaft from the 2,400 level to the 3,600 level horizon for development of the ore zone extending between these two levels.

The Thompson Mine has been in operation since 1961. From that time, until approximately 1980, the mine was primarily a cut-and-fill operation. In the early 80's, vertical block mining (VBM) was introduced in order to:

1) enhance safety by removing miners from the stopes (non-entry mining);

2) to increase productivity and reduce per ton operating costs.

The conversion has progressed to a stage whereby VBM contributes upwards of 80% of the present underground mine production.

VBM is a local term derived from the concept of mining small "vertical" blocks using hydraulically placed backfill (Sarin, 1981). The drilling and blasting of the rock can be accomplished by any means, using small holes, large holes, crater blasting, slash blasting or slot/slash etc. Currently, nearly all ore is broken by crater blasting, though experimentation is continually being done in order to improve efficiencies. Basically, VBM can be considered as a modification of the layout developed for mechanized cut-and-fill stoping; both methods are compatible, with comparable stoping advance per lift.

A general layout is shown in Figure 2. An important distinction is terminology used to describe the excavation. Standard practice at Thompson is to call the immediate excavation a block with a stope consisting of a series of blocks, as illustrated in Figure 3.

Like all mining, VBM at Thompson has evolved over time by a trial and error process. At this particular point, the method has proven to work well. Productivity has increased, together with a greatly improved safety performance. On the other hand, however, though the method has realized a lower per ton operating cost, dilution has increased, resulting in reduced grades. Thus, it is recognized that further modification and optimization are required. The guidelines outlined here will therefore change with time.

A number of constraints exist at Thompson, which influence the application of the method. Great care must therefore be exercised if the Thompson procedures are to be adopted elsewhere. The constraints include:

1) narrow tabular style orebody that pinches and swells; and, a generally weak, schistose hangingwall and footwall;

2) mature mining; therefore existing services sized around initial mining method, with limited mobility for trackless equipment; significant portion of ore tonnage derived from secondary and tertiary mining areas;

3) extended nature of mining; that is many workplaces separated both laterally and vertically with limited access between each for trackless equipment;

4) conversion process.

2.2 Approach

The VBM design guidelines form part of a larger study into VBM stoping at the Thompson Mine. This study has been divided into separate tasks, as follows:

Task 1 - VBM Design Guidelines
Task 2 - Geomechanical/Numerical Model
Task 3 - Blast Design

These tasks are inter-related in their scope and purpose. Blast design will optimize blast performance by improving fragmentation and reducing overbreak. The geomechanical model is designed to determine rock mass strength and reactions to mining. Together, these tasks help to evaluate likely disturbance at excavation boundaries, and this information affects stope design. The design guidelines are then used to produce a final design for VBM stoping. Expected dilution, ground control methods employed, ore recovery, and overall economics are all inter-related in this design process.

In order to develop the VBM design guidelines, a number of individual topics were addressed. Each topic has been reported on separately, and is presented as a stand alone technical report as follows:

Topic	Reported in
Cut-and-Fill Mining	Chapter 1
Data Collection Systems	Chapter 2
VBM Stope Design	Chapter 3
Drilling and Blasting	Chapter 4
Support Requirements	Chapter 5
VBM Stoping Procedures	Chapter 6

Each chapter is essentially self contained and only salient points are outlined in this summary.

3.0 CUT-AND-FILL MINING

3.1 General

The objective of this chapter is to describe cut-and-fill stoping as it is practiced in Manitoba and to examine the potential for improvements, especially of the procedure used at Thompson. Cost and productivities applicable to Manitoba cut-and-fill operations are developed and compared with experience elsewhere.

3.2 Cut-and-Fill in Manitoba

Two cut-and-fill operations are described: the Thompson Mine and the Trout Lake Mine. These mines represent the application of the cut-and-fill method to narrow and wide orebodies respectively. The Thompson Mine typifies a mature, semi-mechanized operation with limited access to the working place. The Trout Lake Mine typifies an operation which has been developed by ramp access from surface and uses mechanized equipment for all aspects of the operation.

3.2.1 Cut-and-Fill Practice at Thompson Mine

The Thompson Mine was developed in the fifties using the horizontal cut-and-fill method in order to achieve maximum extraction of the relatively narrow (ave. 5 m width) ore. Stoping equipment consisted

of jackleg drills, stopers and air-operated slusher hoists. Filling was accomplished by hydraulically-placed classified mill tailings.

Since mining began, a number of important changes have taken place. These include:

- Levels, originally at 61 m (200 foot) intervals, were increased to 122 m (400 foot) in 1972 to decrease the amount of waste development and resultant capital expenditures.
- The uncemented fill used initially has been superseded by a cemented fill; usually at a 30:1 tailings:cement ratio.
- The top 10 m (30 feet) of the cut-and-fill stope was usually extracted by undercut methods. This undercut portion has increased as mining got deeper.
- Slushers have been replaced by 2 cu. yd. loaders.

Conversion to Mechanized Cut-and-Fill was started in 1967. This consisted of replacing slushers with scooptrams and relocating stope ore passes into the footwall. By 1971, twenty-nine scooptrams were operating in Cut-and-Fill stopes and five scooptrams in Underhand Cut-and-Fill pillar stopes. In the majority of stopes, the scooptrams were captive, and could remain in the stope for many years.

Where slushers were used, stope development started at sill heights of up to 10 m (30 ft.) above the elevation of base of rail in the haulage drift. With the advent of scooptrams, stope development started at the elevation of base of rail.

Over the years considerable effort has been put into increasing productivity and decreasing costs. But, within a system where access to a working area is confined to a small raise, there is a limit to what could be achieved. This promoted investigation into other methods and, in 1979, VBM was introduced at the Thompson Mine. Trial stoping proved the value of this technique as a safer, more rapid method of mining. The conversion of Cut-and-Fill stopes to VBM stopes required additional development time and the mine is currently producing 80% of the ore from VBM stopes. The breakdown of mining methods during 1989 are as follows:

Method	Workplaces	Tonnage
VBM	20	82%
Cut-and-Fill	4	14%
Pillar Recovery	5	4%

The production advantages of VBM are evident. It is important, however, to identify the potential for improving the present cut-and-fill system or changing to a different, more mechanized, system.

The Cut-and-Fill system at Thompson, illustrated in Figures 4, 5 and 6, can be divided into seven distinct operations, namely:

- Development
- Opening a New Cut
- Regular Mining Advance
- Excavation Support
- Special Procedures
- Timbering
- Sandfilling.

A description of the more important aspects of these operations is given below:

1) Development

Cut-and-Fill stopes are laid out with strike dimensions, where possible, of approximately 240 m (800 feet). Development begins with the driving of a strike drift in the orebody along the footwall contact at the sill elevation, generally at the base of rail. Widening of the ore sill drift to the hangingwall contact is done by slashing where the orebody is less than about 8 m (25 ft.) wide. Where the orebody becomes wider than about 8 m (25 ft.), additional ground support is achieved by leaving a rib pillar. The back and walls are completely bolted and screened as mining progresses.

A footwall extraction drift or haulage is developed in the footwall rock at a stable distance, usually 10 m (30 ft.) or more, from the stope.

Sill timbering consists of building stringers and a log mat for the recovery of the crown pillar of the stope beneath and the construction of the bases for the service manways. Excavation of boxholes in the extraction drift begins when a minimum of 0.6 m (2 feet) of 10:1 cemented fill has been poured over the log mat. Waste can then be dumped in the stope as fill.

Two bored raises for ventilation are normally driven at the very beginning of the development of a Mechanized Cut-and-Fill stope. The remaining raises are developed as the widening of the stope is completed.

2) Mining Advance

Mining advance is a repetitive cycle, including excavation support, drilling and blasting, advancing services and muck removal. The dominant feature of the Thompson cut-and-fill system is the lack of cycle dependence between drilling and blasting, and ore loading.

Drilling, blasting and excavation support are done off the muckpile. The muckpile is essential as a working platform. In practice a comparatively long muckpile is carried between the face and the ore

loading position. This has the effect of smoothing the influence of LHD non-availability, a significant problem when the LHD is captive.

It is interesting to note how little time, only 12.3 per cent, is spent in drilling and blasting. This indicates the importance of providing alternative faces especially as the LHD utilization is as low as 36% on a single shift basis and 13% for three shifts. Although this importance is recognized, studies have indicated that less than 10% of a stope life is operated with more than one face available.

3) Excavation Support

All roof surfaces are fully supported with expanded shell anchor roof bolts and screen. Walls are also screened down to the limit of the muckpile, depending on conditions. Nearly all of the bolts are 2.5 m (8 foot) in length with a few 4.9 m (16 foot) bolts used when conditions are thought to warrant the longer support.

4) Drilling and Blasting

Drilling is carried out with jacklegs and stopers using 22 mm (7/8 inch) steel with 32 mm (1 1/4 inch) detachable bits. Nearly all the blasting is with Amex and long delay Nonel detonators with the XT cord detonated electrically. Generally, no primer is used.

A typical round in an orebody of average width may contain 18 holes for an advance of 4.6 m (15 ft.) wide by 2.7 m (9 ft.) high and breaking to a length of 2.9 m (9.5 feet) with 3 m (10 ft.) steels. This gives a powder factor of approximately 1.2 kg/m^3 (2.0 lbs/yd^3).

5) Ore Removal

Ore is mucked with 2 cu. yd. scooptrams, mostly Wagner 2B, and tipped into the footwall orepass. There are usually two or more orepasses in a stope, giving a mean one way tramming distance of approximately 40 m (131 ft.).

The average scooptram utilization is 2.9 hrs. per shift. During this period 79 tonnes (87 tons) of ore are trammed resulting in a productivity of 27 tonnes (30 tons) per hour. At a travelling speed of 4 km/hr. a bucket factor 0.85 and swell factor of 0.65 and assuming 50 min. hours then production would be approximately 32 tonnes per hour. Therefore, the achievement of 27 tonnes per hour is reasonable.

6) Productivity

The cut-and-fill stopes at Thompson tend to operate on single shifts with 3 men crews. The average production per stope is in the region of 76 tonnes (84 tons) per shift giving an 'in stope' production of 25 tonnes (28 tons) per manshift. The following gives a breakdown of the time spent per shift on each work element. It is important to note that only 24.4% of the total time is spent in actually breaking and moving the ore.

Work Element	Time Distribution %	Total Man Hours
Ore Breakage	12.3	2.95
Rock Support	20.1	4.82
Ore Loading	12.1	2.90
Timbering	11.6	5.64
Utilities, Etc.	19.0	1.70
Unproductive	24.9	5.98
	100	24.0

3.2.2 Cut-and-Fill Practice at Trout Lake Mine

The Trout Lake Mine is a good example of a modern trackless, Mechanized Cut-and-Fill operation. Information for the following review was extracted from a mine visit and from mine documents (McIntosh and Crocker, 1985).

The Trout Lake orebody, discovered in 1976, is located some 5 km northeast of Flin Flon under the waters of Embury Lake. Figure 7 shows the orebody shape. The ore is hosted by felsic volcanics, overlain by andesites and, further to the hangingwall, argillites. Two types of ore are present, namely massive sulphides and disseminated or stringer type sulphides. There are two main zones being mined at the present time, each of which is made up of a number of lenses. Both zones plunge to the southeast at relatively flat angles ranging from 25-40 degrees. Dips vary from 45 to 65 degrees to the northeast. Widths of the lenses vary from 1.5 to 25 metres and strike lengths from 100 to over 200 metres.

1) Mine Development

Trout Lake was developed as a trackless, Mechanized Cut-and-Fill operation. A single decline measuring 4.2 x 5.5 m (14 x 18 feet) was driven from surface. A second heading, 80 m (265 ft.) below lake surface, was established. These headings were advanced to the ore horizon where both split to service the north and south ore zones, as illustrated in Figure 7. The 80 m (265 ft.) level provides service access to the top of each mining block in the north and south zones for ventilation and fill distribution. Electrical power at 4160 volts is supplied to mine substations through this level.

A ventilation raise to surface was established at the entrance to the 80 m (265 ft.) level. The backfill silos, clear water sump, main pump station and escape raise were also developed in this area. The haulage decline is located 30 m (100 ft.) in the footwall of each ore zone. Ventilation and escape raises were established at the north and south ore extremities.

2) Stope Development

The vertical stoping interval has been established at 60 m (200 ft.). A typical layout is shown in Figures 8 and 9. Stope entry from the haulage ramps is gained centrally along strike at 15 m (50 ft.) vertical intervals starting with the stope undercut. Each entry serves three 5 m (16 ft.) lifts; one at - 16%, one flat, and one at +16%. Entries are backslashed to provide elevation for the next lift.

Stoping begins by establishing the undercut. In the south zone both the footwall lens and the hangingwall lens are undercut. Two ventilation-escape raises are driven in each zone from the undercut to the level above and 127 mm (5 inch) fill holes are drilled down into the undercut. While the undercutting and raising is in progress, a drainage level is established below the stope, leaving a 10 m (33 ft.) sill pillar. To provide rapid drainage of the hydraulic fill, drain raises are driven in the footwall and hangingwall wall lenses at approximately 100 m (330 ft.) intervals on strike. Settling sumps and a pump station are established on each drainage level. Water is pumped to the pump station above. The drainage level also provides top access to the stope below.

3) Mining

From the stope entries, 5 m (15 ft.) high breasts are carried the length of each stope. Work is cycled, whenever possible, so that as one end is being mined the other end is being prepared for filling. Where multiple lenses exist they are carried up simultaneously.

Electric-hydraulic jumbos with 4.3 m (14 ft.) steel and 44 mm (1 3/4 inch) button bits are used for all production and major development. When breasting in a stope, a hole pattern of 1 m (3 ft.) burden and 1.8 m (6 ft.) spacing is used with holes offset on each line. Backholes are drilled on a 760 mm (30 inch) spacing and traced with B line to minimize overbreak. The powder factor is approximately 1.2 kg/m^3 (2.0 lbs/yd^3).

Powder trucks are used to transport and load explosives. An effort is made to use Anfo whenever possible and the mine targets a 4:1 Anfo to stick powder ratio. Nonel is used for priming and all blasts are initiated electrically from designated locations underground. Mine ventilation is such that blasting is permitted at any time during a shift.

4) Productivity

The mine operates on 3 shifts producing 1820 tonnes (2000 tons) per day. The number of people employed at the mine during 1984 was 78. This results in an overall productivity of 23 tonnes (26 tons) per manshift. An estimation of the 'in stope' productivity can be made from the following information:

- 10 direct mining crews produce; 607 tonnes ore, 162 tonnes waste (development),
- sub-development cost approximately $360/m, giving $13/tonne,

- direct stoping cost (less backfill, etc.) is approximately $6/tonne,
- direct mining crew can be distributed between ore and waste on the basis of breaking cost and tonnage broken (ore tonnes = 607, equivalent waste tonnes = 351, giving 6.3 men on ore production, 3.7 men on waste development)
- productivity 'in stope' becomes = 96 Tonnes/Manshift (106 tons)

3.3 Cut-and-Fill Review Internationally

A detailed study of cut-and-fill operations around the world was beyond the scope of this review; the objective being rather to identify modern operations which use alternative procedures or equipment that could improve productivity or decrease costs in Manitoba mines. It was evident early in the review that Trout Lake Mine compares favourably with any operation in terms of equipment, procedures and productivity.

Valid comparisons between operations is difficult as it is often not clear what is included in published production statistics. Also, productivity can vary according to employment practices and labour restrictions. Given the above constraints, the following critical elements of a cut-and-fill operation were reviewed:

- drilling and blasting
- rock support
- ore removal
- backfilling
- stope preparation.

The literature review did not identify any new or unusual procedures which are not already in use in Manitoba, or have not been considered or tested. It did demonstrate that mechanization can result in very large increases in productivity. The common denominator between the various changes which could be made, if the conventional cut-and-fill system is to be improved, is the presence of ramp access to every cut-and-fill face. Ramp access is essential for full mechanization. However, increased productivity does not necessarily result in a commensurate decrease in costs, as the cost of ramp access and equipment has to be matched by a decrease in labour costs.

3.4 Conclusions

Mining methods, procedures and geometries of two very different mines, both operating in Manitoba, have been described and compared in terms of costs and productivities. Results are summarized as follows:

	Item	Thompson	Trout Lake
1	Average Width	5 m	12 m
2	Average Productivity	7 Tonnes/Manshift	23 Tonnes/Manshift
3	In Stope Productivity	25 Tonnes/Manshift	96 Tonnes/Manshift (approximate)
4	In Stope Cost	$20.97/Tonne	$7.28/Tonne
5	In Stope, + Adjustment* for access, Equipment	-	$10.27/Tonne
6	In Stope, + Adjustment* for Orebody Width	-	$15.00/Tonne

*Note: These adjustments are made in order to allow more direct comparison between the two operations.

A brief review of cut-and-fill internationally re ted in the conclusion that:

- Trout Lake Mine was the equal in procedures and productivity of any other operations.
- None of the more modern and highly mechanized operations have captive equipment - they all operate with ramp access,
- There is considerable potential to reduce the powder factor, possibly by 40%.

Nevertheless, the review did not suggest any new systems or procedures which have not already been considered or tried in Manitoba. Likewise, it did not indicate that productivities and achievements are significantly different, given the constraints of methods and orebody geometry.

A fully mechanized system with ramp access was analyzed within the Thompson Mine; constraints and costs were estimated as follows:

- Productivity per Stope = 450 Tonnes/Day
- Labour Productivity (in stope) = 56 Tonnes/Manshift
- Cost per Tonne of Ore = $15.85/Tonne

- Cost per Tonne of Ore with Availability of Faces Reduced by One-Third = $19.75/Tonne

- Cost per Tonne of Ore Increased for Possible Additional Underhand Cut-and-Fill = $21.25/Tonne

In conclusion, the two Manitoba cut-and-fill operations demonstrate a satisfactory level of cost and productivity within the constraints of the methods. It is evident that much higher levels of productivity could be achieved by providing ramp access for the method at Thompson and by increasing mechanization, but it is uncertain whether a decrease in cost could be achieved.

What the study clearly demonstrates is that cut-and-fill remains a viable method, comparable in achievement with VBM in certain geometrical circumstances.

4.0 DATA COLLECTION SYSTEMS

4.1 General

The development and maintenance of a well structured data collection system is of fundamental importance to a mining operation, regardless of the mining method employed. At Thompson Mine, a data collection system was developed based on the case histories from the mining of 126 vertical crater retreat blocks.

The database was separated into two sections: geotechnical and mine operational data. The geotechnical database included categorizations such as geology, structure, strength and rock mass quality. The concept of stress and the effect of the redistribution of stresses through the use of visual monitoring and field instrumentation techniques was included in this geotechnical database.

The mine operational database was separated into categories such as stope geometry, development, drilling, blasting and mucking. The mining efficiency, measured from the combination of parameters from these groupings, was subsequently derived from the mine operational database.

4.2 Description of Database

The Thompson Mine database is set-up on a Microsoft Excel spreadsheet system on a Compaq 386 personal computer. Each VCR block has its own stope data sheet. Data is entered upon the completion of the backfilling cycle of any given block. The information contained on each stope data sheet is automatically extracted into an Excel mine summary spreadsheet where statistical evaluations and correlations can be completed. The process is illustrated in Figure 10.

A total of 81 parameters were collected and 21 parameters calculated for each stoping block, as listed in Table 1. A number of other parameters, for example the results of section sketching, were purposely not incorporated within the spreadsheet database, but rather contained within colour coded stoping plans and sections to be available for subsequent calibration of numerical models.

4.3 Results

Based on information contained within the database, VBM stoping at Thompson Mine can be characterized as follows:

Geotechnical Characteristics

Orebody width:	2-30 m, 9 m ave. (5-100 ft., 30 ft. ave.)
Orebody dip:	45-90°, 69° average
Average mining grade:	2.7% nickel
Mining depth:	150 - 1100 m (500 - 3600 ft.)
Unconfined compressive strength of ore:	70 MPa
Hangingwall rock types:	73% schist 15% quartzite 4% peridotite 8% other
Footwall rock types:	46% schist 34% quartzite 16% peridotite 4% other

Block Geometry

- Height range 15 m-61 m (50-200 ft.), Average height 40 m (130 ft.)
- Width range 2-31 m (5-100 ft.), Average width 6 m (20 ft.)
- Length range 6-40 m (20-130 ft.), Average length 18 m (60 ft.).

Productivities

- Stope tonnage 4,500-63,500 tonnes (5,000-70,000 tons), 18,100 tonnes (20,000 tons) average
- Distance hauled to pass; range 30-210 m (100-700 ft.), Average 104 m (340 ft.).
- Tons mucked per shift; range 180-1,000 tonnes (200-1100 tons), Average 430 tonnes (470 tons).

4.4 Application

Data collection serves three purposes. Firstly, data assists in understanding the actual situation so that proper evaluations can be made. Secondly, the proper evaluation of data will form the basis for actions and decisions. Too often decisions are made on judgement, based on opinions and experiences with no records for backing. Thirdly, a database allows current data to be compared with historical data to determine whether or not a process is proceeding as normal, or whether the process has actually changed.

One important use of the database is the analysis of specific parameters. A distribution of the typical mining block sizes, for example, is a simple yet useful comparison that can be produced from the database in a very short period of time. Based on the 126 block histories, the average block size at Thompson Mine was 40 m (131 ft.) in height, 8.8 m (29 ft.) in width, and 17.4 m (57 ft.) along strike. This may appear to be a relatively obvious set of dimensions that could be reproduced without the aid of a massive database; however, in questioning various engineering and operational personnel regarding the average stope dimension, the number of different answers would vary linearly with the number of individuals asked.

Comparisons that are more complex than the one presented above are just as easily available from the data collection system. For example, the powder factors for production blasting using both 115 mm (4.5 in.) and 165 mm (6.5 in.) holes can be extracted from the database. In both cases, the averages prove to be identical to 2 decimal places (0.51 kg/tonne or 1.02 lb./ton). In absolute theoretical terms, this would not realistically be anticipated, particularly the accuracy to the second decimal place. What the comparison of the distribution of data from the two sizes of hole may indicate, through, is that through trial and error and with time, the charge weights and standoff distances have been adjusted in the field to a point whereby optimum breakage is achieved. This is not to suggest that the powder factors could not be significantly reduced in both cases, particularly given the inefficiencies of crater blasting within a fan pattern of holes. The data does indicate, however, that given the inherent blasting inefficiencies at Thompson Mine, production blasting is in a state of control.

The physical collection of data can also lead to discoveries of potential problems in the mine. For example, collection of geotechnical data in drifts and sills can identify poor ground conditions that can be accounted for in the design of a mining block. Also, visual monitoring can provide an indication of impending problems in current mining areas.

The collection of data can also lead to establishment of levels of control and goals with respect to specific aspects of mine performance. For example, a database will be able to produce a set of representative data with respect to wall dilution. From this distribution, a goal or target can be developed with respect to the upper level of "acceptable" dilution within a given mining block. If this upper level is approached or reached, than a change in mining plan or stope design may be necessary.

4.5 Conclusion

The effective use of a data collection system can be summarized best by the following list of activities:

- comparison of individual mining and geotechnical parameters;
- analysis of operating parameters in order to determine levels of control;
- indication of potential problems in current mining areas;
- development of empirical relationships for the purpose of future mine design.

The cost of establishing such a system may be significant; however, the cost of maintaining it will be minimal given the potential return from the database. As such, the development and maintenance of a data collection system is of paramount importance to the overall success of a mining operation.

5.0 VBM STOPE DESIGN

5.1 General

A major factor in the planning of an underground mine is the choice of suitable excavation dimensions. The economy of size suggests that the larger the excavations, the lower the costs; as larger mechanized equipment can be used, a reduced work force is required. The excavation size, along with the ore geometry is used to determine the most suitable mining sequence. In addition, the design should satisfy the following:

- ensure the overall stability of the complete mine structure, defined by the main ore sources and mined voids, ore remnants and adjacent country rock;
- protect the major service openings throughout their designed duty life;
- provide secure access to safe working places in and around the centres of ore production;
- preserve the mineable condition of unmined ore reserves.

The monetary value of the orebody, and the variation of mineral grade determines both mining strategy and operating practice. The significance of dilution is related to the value per unit weight of ore; marginal ore may become uneconomic if mined by a dilution prone method. Dilution can be reduced with appropriate ground support and careful mining practices, however, there is a limit beyond which additional ground control costs exceed ore worth.

Given the above constraints, an optimum stoping operation is a function of the cost of dilution versus the cost of ground support (which may be in the form of pillars or some form of artificial support). Detailed stope design will depend on a consideration of:

- The size/shape of open stope commensurate with a certain level of dilution and a certain expense for stope wall support and requirement for stope development.
- Extraction sequence.
- Ground support requirements.
- Location of haulage and stope development.

5.2 Thompson Experience

Before discussing procedures for predicting stope dimensioning and sequencing, it is pertinent to review past experience at Thompson. A number of failures of both walls and the back have occurred. These have been the result of several factors including specific geologic structures, excessive stope spans, high stress and overblasting. In terms of frequency hangingwall slough is the most prevalent. Back failure has, however, the most severe consequence.

Though a large database of stoping histories exists (see Section 4.0) little direct information on the size of failure - particularly that in the hangingwall - is available. Rather inferences have to be made from stope performance records. One of the best of these is the difference between the design tonnage and the actual tonnage mined (overpull/underpull). Figure 11 show overpull by block width, while Figure 12 shows overpull by mine area. Overpull, especially when it is accompanied by reduced grade, is generally a result of hangingwall slough which in turn is a result of a combination of blasting practices and stope spans. The fact that the narrower stopes have a higher percentage of overpull than wide stopes underlines the limitations of open stope technique in narrow orebodies.

Based on the above approach, the following statistics have been derived:

Stope hanging walls
- 85 Stable
- 37 Unstable
- 4 Data not available or analysis not applicable

Stope backs
- 90 Stable
- 11 Unstable
- 25 Data not available or analysis not applicable

In an initial attempt to correlate the size of open stope surfaces to their relative stability, a scatter plot of stope hanging wall lengths versus heights was generated and is illustrated in Figure 13. Although there appears to be some correlation in that the larger the stope wall the higher the percentage of failure, it is apparent that additional controls exist. One such control is the duration of stoping. Figure 14 indicates that the likelihood of overpull increases within increased duration of stoping; that is the longer a stope wall is exposed, the more likely it is to fail; this emphasizes the need for a fast cycle time.

Another potential problem area identified through database review, was sandfill stability. Fill walls are required to stand vertical while stoping is underway in the adjacent block. Wall exposure in excess of 45 m (150 ft.) high and 10 m (30 ft.) are not uncommon. Stoping histories, however, indicate a problem with fill stability. Figure 15 compares the amount of overpull in block with a fill wall against those without. Stopes with fill walls, generally experience substantially more overpull than those without, indicating the likelihood of fill slough.

A somewhat different problem is a result of stress build-up in the mining face. Though ultimately this build-up may cause back failure, more typically the first indication will be an increase in the number of blastholes that have to be cleaned or re-drilled and an increase in the amount of oversize at the drawpoint. Sufficient information has been collected in these occurrences to allow such events to be calibrated with predicted level of stress. This is discussed in more detail in the following section.

5.3 Design Methods

Design methods for sizing and sequencing of stoping operations with respect to ground control can be divided into three groups: empirical, analytical, and numerical

Because of the complexity of underground mining no single technique is valid, rather a multi-faceted approach is required. Experience has shown that empirical methods in combination with numerical modelling provides the most realistic predictions.

Empirical Methods

Review of the various available empirical methods indicated that only two were generally applicable to sizing of open stopes; namely the approach proposed by Mathews et al (1981) and the subsequent modification of this approach by Potvin (1988). Both methods use measures of rock quality, induced stress and rock strength to determine the stability of stope backs and walls. The modified design chart is shown in Figure 16.

Based on the level of success achieved in applying these empirical models to the Thompson mine, the following guidelines were established:

1) For the design of future open stope hanging walls at Thompson mine, Mathews' stability graph can be used to predict the potential for wall failure. By no means will the model provide an exact or quantitative result, but it can be used effectively as a guideline in the design process.

2) For the design of future open stope backs, Potvin's modified stability graph displays some validity. For use at the Thompson mine, however, it has been concluded that further calibration of the graph is required. Only 11 open stope back failures were available for initial calibration of the model to the Thompson mine experience. The model at this time can only serve as a very approximate guideline for the design of backs that are mechanically supported by state-of-the-art systems such as cable bolts and cable slings.

3) In order to use these empirical models on a long-term basis, periodic recalibration is required. It is believed that ongoing modifications and improvements to open stoping mining practices will result in an overall decrease in the number of wall and back failures. As future stopes are mined, the empirical database should be expanded and the failure curves adjusted to reflect new information.

Analytical Methods

A number of analytical methods of design were examined including:

- Voussoir arch: analyzing hangingwall behavior using arch theory
- Wedge: Limit equilibrium analysis of joint bounded rigid blocks
- Support interaction: the analysis of drift and support deformations

Though these methods can be instructive in understanding rock behaviour generally, in their present form, they were not found to be particularly useful for stope design.

Numerical Methods

A wide array of numerical methods are available to the stope designer. Of these, the following have been used for ground control studies at Thompson:

- 2-D finite difference (FLAC code)
- 2-D finite element (QUAD code)
- 2-D boundary element (BEM2D code)
- psuedo 3-D displacement discontinuity element (MINTAB and NFOLD code)
- 3-D boundary element (BEEP code)

Because of inherent assumptions with each method (and code), care must be exercised in selecting the method appropriate to the specific problem at hand. The most frequently used method for stope design and sequencing in tabular style orebodies is the psuedo 3-D displacement discontinuity method. Stoping sequences can be modelled with relative ease and the change in induced stress within the ore horizon predicted. Experience has shown that the method provides reasonable result for back and pillar stress but is of limited use when trying to examine hangingwall or footwall behaviour. Regardless of the numerical method used, it is critical to calibrate the predicted results with reality; that is the events that are actually occurring underground. This calibration is fundamental to any use of numerical modelling in mining. Based on results contained within the stoping history database, (visual monitoring, section sketching and instrumentation) calibration of stresses predicted by MINTAB has been carried out, as given in Table 2.

5.4 Conclusions

A philosophy of stope design has been developed in which past experience is used to calibrate both empirical and numerical models. In turn, these models are used to assess applicable stope dimensions and to predict ground control events associated with different stoping sequences. This must be done within the framewok of ore value, tonnage and prediction requirements. The philosophy adopted is outlined in Figure 17.

6.0 DRILLING AND BLASTING

6.1 Practice

The technique of crater retreat stoping was developed from 1974 onwards at INCO'S Sudbury Basin mines in conjunction with Canadian Industries Limited (CIL). It was a response to the high cost of existing methods of pillar recovery and its use was extended to slot development for sublevel open stopes and as a replacement for cut and fill and post pillar mining. It is now favoured as a primary stoping method at both Sudbury and Thompson. Benchmark papers prepared by Lang, et al (1977 & 1978) described some of the work carried out at the INCO mines at Sudbury, the Birchtree mine near Thompson, Centennial Mine at Flin Flon, Heath Steele Mines at Newcastle, Sherrit Gordon Mines at Lynn Lake, Whitehorse Copper Mines at Whitehorse and Pamour Porcupine Mines, Porcupine. Other applications described were carried out at Rubiales, Spain and at Ardlethan, Australia.

Large scale VCR-type blasting operations in deeper mining that can be usefully compared with Thompson are carried out for the Homestake Mine at Lead, South Dakota, O'Okiep copper mine in South Africa and at Cobar Mines in Australia.

The majority of VBM stopes at Thompson are converted from cut and fill stopes. Minimum stoping width is about 4 m (12 ft.) and stopes up to about 12 m (40 ft.) wide are mined longitudinally; wider stopes are mined transversely (Figure 18). Stopes wider than 8 m (25 ft.) are mucked through transverse drawpoints developed just below the back of the converted cut and fill stope. New VBM areas are usually developed with a footwall ramp to connect all sublevels.

Raise and slot development in VBM stoping is eliminated since the undercut provides the free face for blasting. The drill drive (overcut) is usually about 5 m (15 ft.) wide, hence holes are drilled in an inverted fan pattern, with toe spacing up to ten times that of collar spacing to cover stoping widths ranging from about 4 m (12 ft.) to 12 m (40 ft.). Fan drill patterns must also accommodate irregular ore/waste contacts, with drilling angles ranging from the vertical to approximately 55 deg. from horizontal. Ideally, all drill holes should breakthrough to the undercut, but in practice, many of the perimeter holes are "blind" and are fired initially as slash holes. Blastholes are drilled with ITH (in the hole) hammer drills. Hole diameters between 114 mm (4.5 in.) and 165 mm (6.5 in.) are used in production stopes. Some statistics are as follows:

HOLE SIZE	TONS BROKEN PER METRE DRILLED		METRES DRILLED PER SHIFT	
	Average	Coeff.Var^{n} (%)	Average	Coeff. Var^{n} (%)
114 mm dia.	0.8	>100	35	66
165 mm dia.	1	>100	32	100

6.2 Loading and Blasting Procedures

The first step is to inspect all of the holes and ream blocked or partly closed holes required for the blast. The second step is to profile both the back of the stope and the muckpile. These measurements are critical, since they determine charge placement, firing sequence (expansion room) and provide a record of breakage for each blast cycle.

The charge configuration and geometry for 114 mm (4.5 in.) diameter and 165 mm (6.5 in.) diameter blastholes are given in Figures 19 and 20. Wooden blocks are lowered down the hole to a predetermined depth and tied off. Two or three capsules of stemming are then dropped to ensure that the hole is properly blocked.

ANFO and Aquamex (watergel) are the commonly used explosives. Standard charge weights are 2 x 12 kg (27.5 lb.) cartridges for 165 mm (6.5 in.) diameter holes and 2 or 3 x 2 kg (9.0 lb.) cartridges for 114 mm (4 in.) diameter holes. The explosive charge is centrally primed and initiated either by a down-the-hole delay system with a 450 gm slider primer and 5.3 gm/m detonating cord or collar delays with a 230 gm booster in association with 5.3 gm/m or 10.6 gm/m detonating cord. Detonators are short period nonels. Those parts of the explosive charge which enter the blasthole before and after the primer, are dropped from the collar to obtain maximum slump. Approximately 3 m (10 ft.) of stemming (usually mill tailing) is added to confine the charge.

Multiple charges in one blasthole are usually 1 m (3 ft.) long, spaced 3 m (10 ft.) centre to centre and separated by mill tailing. Deck charges are usually initiated by the down-the-hole delay system.

Statistics available for powder factor indices are presented below.

HOLE SIZE DIA. (mm)	NUMBER VBM BLOCKS	POWDER FACTOR (Kg/TONNE) Average	Coeff. Var^{n} (%)	RANGE (Kg/TONNE)
114	43	0.5	84	.3 - 1.0
165	79	0.5	88	.2 - 1.0

6.3 Experience

An average VBM block at Thompson only contains about 18,140 tonnes (20,000 tons) of ore. By industry standards, these are small tonnage blocks to apply VCR blasting techniques over high lifts (average 40 m (130 ft.)). Wall damage is inevitable with the high powder factors used (average 0.5 kg/tonne (1.0 lbs./ton)) to blast ore bounded by weak schists. The average depth of slough beyond planned stoping limits is estimated to be 1.5 m - 3.0 m (5 ft.-10 ft.) for 27% of the VBM stopes mined to date.

The average powder factor at Thompson is similar to the industry standard for breaking 3.0-3.6 m (10-12 ft.) slots in conventional longhole stoping. However, full column slash blasting applied to the average stoping width of 9 m (29 ft.) should permit powder factors of 0.25 kg/tonne (0.5 lb./ton) using ANFO and much less using diluted ANFO. Heavy blasting combined with higher ground stresses associated with remnant and deeper mining only point to ever increasing dilution and ore loss with time, unless the amount of ore broken by VCR and associated deck blasting techniques is decreased appropriately.

The greatest challenge lies in reducing the dilution associated with the mining of narrow, sinuous ore bodies (less than 6 m (20 ft.) thick) while maintaining the productivity and costs currently being achieved. Two case examples prepared by mine staff, the 1403L - 301 VBM stope and the 3600L - 231 VBM stope, highlight the problems.

The hangingwall schist, over the 366 m (1200 ft.) strike length of the 1403L - 301 VBM stope, can be described as fair (RQD greater than 50). Average ore width was 3.4 m (11.3 ft.), (range from 2 - 7.0 m (6 ft.- 23 ft.)), but the mined width averaged 5.3 m (17.2 ft.), yielding an average dilution by volume of 52%. Additionally, 20-25% of the ore reserve within planned mining limits was lost in undrawn ore, frozen remnants from deck blasting and irrecoverable and fractured rib pillars. The average data on blasting is given below:

- Blasthole size 114 mm (4.5 in.) diameter with area of influence, 1.97 sq. m (19.1 sq. ft.).
- Powder factor, 0.45 kg/tonne (0.9 lbs./ton).

- Break per crater blast, 1.92 m (6.3 ft.).

Difficult charging conditions were experienced in at least 5 blocks, at locations where the induced stress levels, estimated from the results of numerical modelling, exceeded 35 MPa (see Table 2). Poor drilling and blasting conditions were experienced in the last two blocks mined at locations where estimated induced stress levels exceeded 45 MPa. Narrower than expected ore at the base of one block resulted in too few breakthrough holes. Very heavy blasting near the brow exposed schist, which then sloughed back to quartzite.

The rocks over the 153 m (500 ft.) strike length of 3600L - 231 VBM stope can be described as poor (RQD less than 50). Four stopes were mined over a height of about 24 m (80 ft.), with two stopes each side of a central 18 m (60 ft.) rib pillar. One of the stopes next to the rib pillar was about 6 m (20 ft.) thick, but the remainder were quite narrow at 3-3.6 m (10-12 ft.) thick. Drilling and blasting in the thick stope was problem free, but severe hole closure occurred in the narrow stopes, resulting in difficult charging, frozen blasts and lost ore. Only 50% of the holes were available for blasting in one stope. Compared with 1403L - 301 VBM stope, the rocks were of lesser quality, but mining was 671 m (2,200 ft.) deeper and induced stresses were high enough to close blastholes adjacent to narrow, 3-3.6 m (10-12 ft.) abutments. Hence there is a potential for severe problems associated with longhole blasting in deep, narrow ore.

Based on past experience, ground control problems are most severe when opening up a new block. Conditions tend to improve as the mining sequence is established. The reasons for these occurrences need to be understood. There is a possible need for improved ground support in the early stages of mining, but more importantly, enforcement of the minimum 3-2-3 pattern of breakthrough blastholes into the undercut may be necessary if VCR blasting is to remain a major method of ore breaking in narrow orebodies. Changes to this pattern may be used if a modified blasting technique such as crater-slash is used.

Considering the mining of thick orebodies, there should be few problems associated with the closure of blastholes due to induced stresses. If problems do occur at some locations, they can probably be resolved by adopting a layout which involves the mining of an expansion slot followed by a mass blast of the remainder. An example of thick ore body stoping monitored by the mine staff is the 1800L - 330 Transverse VBM stope. In this case, sufficient ground was undercut to cause back failure (possibly triggered by deck blasting practices). It seems that ground control would have been improved with good quality cemented fill (1 to 1.7 MPa (150-250 psi) uniaxial strength) combined with timely cable support and modified deck blasting practices. There were no reports of closure of blastholes due to induced stresses.

6.4 Design Guidelines for Drilling and Blasting

Currently, nearly all of the ore in VBM stopes is broken using conventional VCR blasting techniques. Supplemental multi-deck blasting procedures are used to expand blocks laterally by slashing and to recover crowns. It has been shown that the cost advantages and higher mining efficiencies

(compared with mechanized cut and fill stoping), have resulted from the application of VCR blasting techniques. However, these advantages have been offset in part by very heavy dilution in narrow, sinuous orebodies and reduced ore recovery, compared with traditional mining and/or rock breakage approaches. As noted previously, this is a conscious trade off, but it seems that the original concept of VBM as defined earlier is still appropriate, particularly the observation relating to the application of drilling and blasting practices. It is beyond the scope of this report to detail alternative rock breakage approaches that may be applicable, but it is suggested that full column bench blasting has a future role in stoping operations. The remaining discussion is focused mainly on guidelines for current practices, considering rock breakage, fragmentation and damage aspects.

Blasthole Drilling Practice

- Blastholes should be drilled parallel to the hangingwall. Any fan holes drilled should intersect the footwall only.
- Three blastholes per ring should break through into the undercut, with a minimum of two breakthroughs.
- Break through holes should be surveyed and tied back to their collars; the information should be used to revise the blast design as appropriate, including the drilling of additional blastholes as needed.
- The portion of blastholes drilled beyond design wall limits should be plugged 1 - 1.2 m (3 - 4 ft.) inside the design wall limits.
- Considering damage, 114 mm (4.5 in.) blastholes should have preference over 165 mm (6.5 in.) blastholes.

Blasting Practice

- The powder factor should be reduced to match rock condition. Aim for a powder factor of less than 0.22 kg/tonne (0.5 lb/ton) for VCR blasting operations in narrow orebodies, based on the use of ANFO or equivalent explosive.
- Eliminate current deck blasting practices to reduce damage. Use full column charging with diluted ANFO to bench the crown after establishing a slot. Reduce hole diameter by artificial means to further reduce damage if necessary.
- Test the method used at the Homestake mine to extract about one third of a stope block by conventional VCR blasting and the remainder in a continuous full column bench blast using diluted ANFO.

- Standardize drilling and blasting practices throughout the mine using specialist supervision as practiced at the Cobar mine.

- Reduce explosive consumption by determining the limiting range of charge spacing needed to avoid charge malfunction or dislodgement.

- Monitor blasting practices regularly to maintain standards and test new procedures.

7.0 SUPPORT REQUIREMENTS

7.1 General

Tentative guidelines for the selection and design of support in development openings have been developed. These guidelines are based on current industry practice, accepted design procedures and experience gained at Inco's Thompson Mine.

The need for simple guidelines for ground support design in underground mining has become evident during the last decade. This is largely a result of two factors: first, the trend toward high production, bulk mining systems has increased the demands that are placed on ground support; while secondly, there has been several new and innovative support techniques developed. In contrast, there has been little revision to ground support design procedures and practical education of mine operators as to the impact of the new methods. This has led to misuse and non utilization of support techniques, resulting in lost opportunities for more productive mining.

Support of mine openings is part of the daily operations of all underground mines in Manitoba and throughout Canada. The purpose of support can be simply stated as the need to keep excavations open and safe for their required life span. This can be achieved by the design of relatively stable excavations, and by the installation of internal and external artificial reinforcement.

Costs associated with support, or conversely no support, can be substantial. For example, support may account for 25 per cent of the total cost of developing an access drift. If inadequately designed for actual ground condition, and more importantly, change of ground conditions, costly rehabilitation may become necessary.

Though support, or lack of support, can be a significant cost centre, little design is generally undertaken. Rather, the premise of "what has worked before" is relied upon. Also, little quality assurance is carried out during support installation or afterwards, during the functional life of the opening.

The design of rock support is dependent on a large number of factors including:

- rock quality (strength, deformability, jointing, etc.);
- prevailing stress;
- change in stress due to adjacent mining;

- size of opening;
- function of opening (e.g., drill drift, drawpoint, stope, etc.);
- consequences of rock behaviour;
- properties of support; and
- cost of support and, conversely potential costs of not supporting.

Because of the complex interactions between these parameters, no single design equation is feasible. A considerable amount of judgement is required based on prior experience and the results of a variety of empirical, analytical and numerical models. Good judgement provides the key to a successful pragmatic design.

The objective of the support guidelines proposed is to provide a rational basis for selecting support type and density to ameliorate those problems that have the greatest impact on mining, that is back failure, access loss or deterioration and, hangingwall slough. Though primarily oriented towards development excavation support, the guidelines also address the requirements for stope walls.

Any procedure for support design and selection must be largely empirical due to the many variables involved. Five steps can be identified:

1) Characterization of rock mass
2) Prediction of induced stress over life of opening
3) Prediction of likely ground behaviour/failure
4) Support options available
5) Selection

These guidelines therefore provide procedures associated with steps 1 to 4, while the final step, selection, must remain with the user.

7.2 Support Design Methods

The behaviour of reinforced rock masses is not completely understood. This is due, in part, to the difficulty in characterizing the mechanical properties of a rock mass which is a discontinuous and heterogeneous material and, also accounting for the changing load conditions which occur as mining progresses. Current technology cannot accurately determine support requirements. As such, any design method should be used as a guide only.

Four methods of design are used: industry experience, empirical methods, analytical methods and numerical methods.

Industry Experience

Existing mining operations contain a wealth of information on support systems. In most cases, these systems have evolved through a trial and error process to the point that they are suitable to the needs

of the particular mine. Though each mine has its own specific ground conditions and operational constraints, experience can be transferred from one mine to another, provided that due care is taken.

Ground support practices which are currently in use are summarized in tabular form in an appendix of Chapter 5. This data encompasses a broad range of operational experience at mines in Canada, Australia, Sweden and South Africa and forms the basis of support design based on industry experience. The currently accepted methdology for cable bolt design is also described, having been totally derived from industry experience.

Empirical Methods

Empirical methods of design are based on observations of rock mass and support behaviour in similar situations. Rock mass classification systems, excavation size, and unstable block conditions are used as a basis for these methods. The methods were developed for use in civil excavations where geometry is typically much simpler than in mines; as such, due care must be taken.

Many simplifying assumptions are inherent with the approach. It is generally accepted that the empirical design methods provide a good starting point for support design, provided that sound engineering judgement is used. These assessments are most useful where preliminary design is necessary but only limited geotechnical data is available. The rock mass classification systems themselves provide a convenient way of comparing different rock mass conditions and relating support design experience. Barton's Q System and Laubscher's MRMR System are described in detail, while other schemes are referenced and briefly described.

Analytical Methods

Analytical methods are used for design of support where structurally controlled failure is anticipated. The methods do not account for stress induced failures. These methods of structural stability analysis can be divided into three groups; limited rock block stability, beam theory and arch theory.

The Limit Equilibrium Stability Analysis determines support requirements for wedges or blocks that are free to fall or slide under their own weight. This involves determination of the block shape and size, and whether or not kinematic failure is possible. The Voussoir beam theory is applicable to situations where one set of discontinuities dominates and trends parallels the excavation surface (e.g. foliation along the hangingwall). Arch theory is a more generalized version of the voussoir beam theory. The formation of natural arches above underground excavations is observed in most mines, and are the result of the stress redistribution about the opening.

Numerical Methods

Numerical methods to predict stresses around excavations have advanced significantly over the past few years. A number of techniques are in use, the most common being finite element, boundary element, distinct element and finite difference. Rock bolts and discontinuities can be represented in these

models by special elements. These elements behave in a theoretical manner, as do all the elements in the models. In the real world, the variable nature of the geology, and timing and quality of support installations make it impossible to predict support loads and stresses with a high degree of accuracy.

An important advantage of numerical models is the ability to do comparative studies of different excavation and support designs. These studies can be done at relatively low cost in the pre-production stage. Analysis can be made of a number of variables to determine the influence on design and thus arrive at a theoretical optimum. The results of these models are very dependent on the accuracy of the input parameters. Models should be calibrated by reference to monitored in situ conditions.

It is foreseen that numerical methods will continue to be advanced and refined. The use of such models is likely to become an ever more valuable tool in the design of rock reinforcement.

7.3 Support Design at Thompson

Standard practices currently in use for ground support at Thompson are 2.4 m (8 ft) mechanical bolts (expansion shell anchors) on a 1.2 m x 1.2 m pattern with 102 mm (4 inch) No. 6 and No. 9 gauge screen for all underground development. In addition, the following support systems are used:

Rockbolting:	- 1.8 m (6 ft) Split Sets on 1.2 m x 1.2 m pattern - 4.9 m (16 ft) mechanical bolts in ore extraction drifts on 3 m (10 ft) spacing - cable slings used in walls with very slabby ground
Grouted Anchors:	- 6.1 m (20 ft) cable bolts in drill drift sills on 2.4 m (8 ft) row spacing, 5 per row - 12.2 m (40 ft) cable bolts in some places
Shotcrete:	- 50 mm (2 inch) thick in drill drift sills and waste drifts with poor ground

The maximum span for development openings in ore is currently set at 6.1 m (20 ft). In thicker ore zones, multiple drill drifts are generally developed rather than silling out wider than 6.1 m and using extra ground support. However, the potential exists to increase this maximum span through incorporation of more extensive support. This would benefit drilling and blasting performance.

The proposed guidelines for support design at Thompson Mine follow a linear path with input from various data bases and computational methods. The following is a flow chart of this proposed path.

EXCAVATION CONDITIONS
size
shape
intended use (acceptable factor of safety)
geology
rock mass quality

DEPTH
determine virgin stress

INDUCED STRESS HISTORY
reference to data base
separate numerical analysis

MODE OF FAILURE
identify stress conditions (compressive, tensile)
identify specific structures (wedges, thin laminae)
identify failure mode (buckling, shear, pop out, gravity)
identify size of failure

SUPPORT TYPE SELECTION
based on dead weight requirements
based on interaction with rock

SUPPORT DESIGN
based on dead weight requirements (block and wedge)
based on previous experience
based on rock mass quality

Rock mass quality is determined through the empirical methods described previously. Numerical methods are used to estimate induced stress around mine openings. Analytical methods are used to determine the actual support requirements based on the most likely failure modes. Industry and local mine site experience is generally referenced to see what has and has not worked in similar conditions.

In the more general situations support will be based upon a quality classification of the rock. Experience suggests the 'Q' method as the most suitable for use at Thompson. However, experience has also shown that support will depend on the ratio of maximum boundary stress in the opening to the compressive strength of intact rock. This fact has been observed elsewhere, and data has been tabulated into a design chart by Hoek (1980). The experience at Thompson has been incorporated into this chart, and the results are presented in Figure 21. This design chart permits selection of ground support based on rock mass quality, intact rock strength and induced boundary stress. Potential users are warned that this chart should only be used for preliminary assessment of the ground support of proposed excavations.

The support requirements guidelines illustrate practical applications for determination of ground support requirements in underground mine openings. These guidelines are based on the collective experience gained at Thompson and other mining operations throughout the world. In terms of VBM stoping, they are slanted toward support of development openings and stope backs. Support for stope walls is not specifically referenced because walls should be supported by blasted ore in the stope during shrinkage draw. Stability of stope walls is covered in more detail in Chapter 3, VBM Stope Design.

8.0 VBM STOPING PROCEDURES

8.1 General Procedures

Chapter 6 describes stoping procedures for vertical block mining (VBM) as they are practiced at the Thompson Mine. A description of stoping procedures at other similar operations is also provided to illustrate practical mining solutions which have evolved in somewhat different geologic environments. These operations are the Homestake Mine in USA, Carolusberg Mine in South Africa and Cobar Mines in Australia. The goal of this work is to provide workable solutions to problems which will likely be encountered during the implementation of VBM stoping in a steeply dipping orebody in any hard rock mining environment.

VBM mining requires the development of an overcut drill drive and an undercut mucking level for longitudinal mining. Drawpoints and a haulage drift are required for transverse mining. Raise and slot development are eliminated because the undercut provides the free face for blasting.

Minimum drill drift dimensions are 3.6 m (12 ft.) wide x 3.6 m (12 ft.) high to accommodate ITH drills. Wider overcuts are developed where ground conditions permit in order to obtain a greater degree of parallel hole drilling. Single drill drifts are generally driven along the hangingwall contact. Where orebody width is excessive, two roughly parallel drill drifts can be driven, with one along each contact.

The extraction drift is generally driven in footwall rock, up to 30 m (100 ft.) from the orebody. Previously, drifts were much closer to the stoping areas, but major problems with stability were experienced. Thus, to avoid costly rehabilitation, the decision was made to locate the drifts at a greater distance into the footwall. Drawpoints are driven off the extraction drift into each block. Where hangingwall rock is more competent, the extraction drift is sometimes driven in the hangingwall.

Large VBM stoping areas are serviced by a ramp access to facilitate movement of mobile equipment and supplies. Ramps are driven at a maximum grade of 20%, and connect all sublevels.

VBM stope dimensions are determined by the prevailing ground conditions, stress levels and orebody geometry. The minimum stoping width is about 3.6 m (12 ft.) Stopes with widths up to 12 m (40 ft.) are mined longitudinally (Figure 22). Wider stopes are mined transversely. Apart from these general guidelines, there are numerous variations to the extraction method imposed by structural and geometrical considerations.

The first VCR blast in a stope utilizes the undercut drift for expansion room. As the blasting progresses upwards, sufficient swell is mucked to maintain expansion room for subsequent blasts. Non-

breakthrough holes are slashed (using crater charges) into the void when the breakthrough holes have been VCR blasted to a sufficient height. The blasting and mucking cycle is repeated until 8 m (25 ft.) of ore is left below the overcut drill drift. This ground is taken out in one blast using decked charges separated by stemming.

The blast firer is responsible for ensuring that all affected areas are cleared and guarded before the blast is fired. A blasting letter is issued by the Engineering Department for all blasts in excess of 900 kg (2,000 lb.) of explosive, detailing ventilation arrangements and any other precautions to be taken. Generally, all major blasts are conducted at midnight on a Friday to permit exhausting of fumes over the weekend.

Mucking at Thompson is carried out with 1.5-4.6 cu. m (2 to 6 cu. yd.) LHD's. Ore is mucked through drawpoints (transverse retreat) or within the stope (longitudinal retreat). Ideally, during blasting cycles, the stope should be kept full of broken ore to provide wall support, allowing LHD's to muck ore without entering the stope. However, this ideal is rarely achieved due to production constraints. The end result can be excessive dilution from the hangingwall and adjacent backfilled blocks, and the need for a large percentage of the substantially slower remote control mucking.

General procedures for crown pillar recovery and backfilling are described in detail in Chapter 6. For backfilling, several figures are included which illustrate general arrangements of barricades and drainage facilities in VBM stopes and barricade construction is in detail.

Stoping procedures for Thompson and the other three comparison mines are summarized in an appendix of Chapter 6. The procedures for Thompson are summarized below.

Layout

Overcuts:	Drift size	=	37 m (12 ft.) wide x 37 m (12 ft.) high minimum
	Support	=	screened and bolted with 2.4 m (8 ft.) mechanical bolts. Split sets and 6.1 - 12.1 m (20 - 40 ft.) cable bolts depending on width and ground conditions.

Drawpoint, extraction drift, ramp and ore pass access drift dimensions vary with size of LHD used, as follows:

2 cu. yd. LHD	=	3.1 m wide x 3.2 m high (10 ft. x 10.5 ft.) minimum
3.5 cu. yd. LHD	=	3.7 m wide x 3.2 m high (12 ft. x 10.5 ft.) minimum
5 or 6 cu. yd. LHD	=	4.0 m wide x 3.2 m high (13 ft. x 10.5 ft.) minimum
Drawpoint support	=	rebar, with screen.

Drilling and Blasting

Hole size:	114 mm and 165 mm (4.5 and 6.5 inch) diameter.
Drill pattern:	Drilling from overcut drift. Parallel hole drilling if possible, but normally inverted fan pattern with many perimeter holes non breakthrough.
Hole spacing:	1.5 m spacing, 2.1 m burden for 114 mm holes (5 ft. spacing, 7 ft. burden for 4.5 in. holes). 2.4 m spacing, 3.1 m burden for 165 mm holes (8 ft. spacing, 10 ft. burden for 6.5 in. holes).
Depth of charge burial:	1.2 m for 114 mm holes (4.0 ft. for 4.5 in. holes). 1.8 m for 165 mm (6.0 ft. for 6.5 in. holes).
Explosive type:	ANFO or packaged emulsion/slurry.
Charge weight:	8.1 kg for 114 mm (18 lb. for 4.5 in. holes). 25 kg for 165 mm (55 lb. for 6.5 in. holes).
Average advance rate:	2.1 m per blast for 114 mm (7 ft. per blast for 4.5 in. holes). 3.1 m per blast for 165 mm holes (10 ft. per blast for 6.5 in. holes).
Powder factor:	variable, 0.5 kg/tonne (1.02 lb./ton) ave.
Crown pillar recovery:	stop VCR when 7.6 m (25 ft.) of ore remaining, load remaining holes with multiple charges and blast in one shot.

Ore Handling

Mucking:	1.5, 2.7, 3.8 and 4.6 m^3 LHD's (2, 3.5, 5 and 6 cu. yd. LHD's), diesel powered
Expansion void height:	1.5 times blast volume (normally 2.7 m (9 feet) minimum)

8.2 Comparison with Cut-and-Fill

The increasing use of VBM over cut-and-fill stoping at Thompson is proof of its many advantages. Although cut-and-fill provides greater flexibility, VBM provides significantly higher productivities and lower cost per ton. The economic advantages of VBM are considerable even when the extra cost for development (but not dilution) is taken into account. Experience at Thompson has shown that the direct mining cost per ton, including all required development, is 20% less for VBM than for cut-and-fill (Ludwig, 1985). The advantage in productivity is more dramatic. Typical values in pillar recovery are 10 tonnes/manshift (12 tons/manshift) for cut-and-fill and 47 tonnes/manshift (52 tons/manshift) for VBM (Stewart, 1988). The impact of the difference in productivities on a time-based cash flow analysis is significant.

The major disadvantage of VBM is its inability to follow irregular ore geometries, and the requirement for overhead pillar protection to the drilling horizon. Ore recovery has been reduced by as much as one third in some stopes. Cut-and-fill mining will normally provide greater ore recovery due to its inherent flexibility.

In terms of manhours, cut-and-fill will require about 60% more manhours than VBM to mine a given tonnage. The distribution of manhours by activity is as follows:

Activity	Cut-and-Fill	VBM (ramp access)
Production	37.1%	41.5%
Ground Support	19.6%	9.6%
Miscellaneous	19.2%	14.1%
Maintenance	11.2%	14.9%
Development	8.5%	13%
Fill Preparation	4.4%	6.9%

It should be noted that although a percentage given for cut and fill may be lower than the corresponding number under VBM (such as for fill preparation), the actuai time required for the task may be higher because VBM requires less time overall to mine a given tonnage.

There are distinct differences between methods of grade and production control for cut-and-fill and VBM mining. In VBM total reliance is placed on diamond drilling information, whereas cut-and-fill places only partial reliance on diamond drilling due to its ability to follow unexpected orebody geometries. Methods of production control for VBM mining are summarized as follows, with those for cut-and-fill included for comparison:

Activity	VBM	Cut-and-Fill
Diamond drilling	Total reliance	Partial reliance
Orebody outlines	By interpretation	From cut mapping
Geological mapping	Sills mapped, minimum of 24 m (80 ft.) intervals	Backs mapped, 3.1 m (10 ft.) intervals
Muckpile monitoring	At drawpoint	In stope
Dilution estimation	By observation only	Survey pick-up in stope
Sand dilution	Incidental	Pre-determined
Production rates (for similar size stopes)	430 tonnes/day (470 tons/day)	90 tonnes/day (100 tons/day)

A critical component of the economics of VBM stoping is dilution. Potential sources are wall rock failure and sand from a neighbouring block. Experience at Thompson has shown that hangingwall and back failures are the predominant sources of dilution. Statistics from ground control studies have indicated hangingwall failure in at least 31% and back failures in at least 11% of all VBM blocks mined.

Hangingwall slough is normally approximated through observation of the stope walls when mining has been completed, and infrequently with downhole extensometers or cables. Estimations typically range up to 3.6 m (12 ft.) with two extreme values of 6.1 m (20 ft.) and 7.6 m (25 ft.) The actual amount of rock missing off the walls is a judgement call which is difficult to make. These figures are thus approximations.

Hangingwall failure has been correlated with a number of parameters in efforts to determine its contributing factors, including powder factor, rock type, block dimensions and rock quality. Undoubtedly all of these will have some effect on dilution. The strongest correlation appears to be with rock type, particularly with the presence of a biotite schist in the hangingwall. This schist ranges in thickness from 1.5-7.6 m (5 to 25 ft.). In a total of 94 blocks with a schist hangingwall, failure occurred 34% of the time at an average slough of 2.1 m (7 ft.). Other rock types occur much less frequently and result in less hangingwall slough when they do occur.

Dilution is a serious problem in VBM stoping because VBM workplaces don't have a built-in ability to sort and store dilution as it is recognized. Some workplaces have limited storage for waste rock, but normally everything in the drawpoint is mucked as ore.

The economic consequences of dilution can be illustrated with the following example. Using the direct mining costs (including development) costs quoted by Ludwig (1985), and assuming an ore grade of 2.7% Ni, the following two cases represent identical mining costs per pound of nickel:

Cut and Fill: 10% dilution (assumed)

VBM: 36% dilution (calculated)

Hence, the choice between VBM and Cut-and-Fill must also consider realistic estimates of the expected dilution. The economic advantages of VBM can be quickly lost if dilution is not controlled.

8.3 Comparison with Other Mines

The VBM method has been successfully used at Thompson and at a number of other mines worldwide. A common thread between the mines is the use of VBM to convert from cut-and-fill methods to more productive bulk mining or, to convert areas experiencing significant ground control problems.

Table 3 summarizes key statistics from the Thompson Mine and three other operations selected for comparison. Perhaps the most outstanding difference between Thompson and these other mines is average stope tonnage. Typically, Thompson will draw about 18,140 tonnes (20,000 tons) of ore from a stope, compared with the 90,700 tonnes (100,000 tons) drawn at the other mines. This emphasizes the application of VBM at Thompson to both wide and narrow orebodies, while at the three other mines, VBM

tends to be used only in wider sections of their orebodies. Homestake did try mining narrow orebodies, but found that though productivities were comparable with wider orebodies, dilution (primarily a result of hangingwall blast damage) was very high. Thompson has, for the time being, accepted this trade-off between increased productivity and increased dilution associated with mining narrower orebodies.

Another area of difference is in the use of a shrink draw. The tendency at Thompson is to pull stopes empty after blasting each lift, whereas the other mines will operate a shrink draw to provide maximum hangingwall support during mucking. Though the benefit of a shrink draw is strongly endorsed at Thompson, the reality of operating a mature mine with many scattered working places, and poor access for trackless equipment between the workplaces (a function of the conversion) results in severe production demands from individual stopes.

At Thompson, the VBM method has been used in conjunction with crater blasting techniques. Ideally, this approach is suited for wide orebodies (greater than 7.6 m (25 ft.) width) with strong competent footwalls and hangingwalls. Thompson have been successful in applying the method in far from ideal circumstances, that is in narrow orebodies with weak hangingwalls, in order to achieve major productivity gains. The trade-off has, however, been increased dilution. Because of the grade of the Thompson orebody and historical nickel prices, this trade off has generally been acceptable. However, recent high nickel prices have substantially increased the cost of dilution, and the mine has embarked on a program to further investigate the causes and adjustments required to the VBM method in order to minimize the amount of dilution.

9.0 CONCLUSIONS

A number of studies have been undertaken to investigate the application of VBM at Inco's Thompson Mine in Manitoba. A set of design guidelines have thus been produced to provide a rational basis for mine design, with particular reference to mining of steeply-dipping orebodies.

Potential users of these guidelines are cautioned that Thompson is a mature mine which has been converted from the original mining method for which it was developed. Portions of the manual may therefore not be applicable at other mines.

Overall, the studies have shown that VBM can be implemented in narrow steeply-dipping orebodies, and that major productivity gains can be realized over cut-and-fill techniques. However, higher dilution rates must be tolerated, particularly where weak hangingwall and footwal rocks are present.

The importance of data collection has been reaffirmed by this work. Data collection formed the cornerstone of the study and provided a basis from which all recommendations were made. It provided the only means of monitoring stoping performance, thus contributing to ongoing efforts to optimize and improve on results obtained. Decisions for future work could therefore be made based on fact rather than opinions.

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TABLE 1: DATA COLLECTION PARAMETERS

DATA BASE CATEGORY	PARAMETERS	SOURCE	COMMENT
GEOTECHNICAL CHARACTERISTICS	RQD	Scanline Survey Core Logging	Result corrected for bias
	Ja, Jn, Jr	As Above	Parameters for rating rock according to Q System of Barton et al (1974)
	Q	Calculated	
	Primary Rock Type!	Geologic Mapping	
	UCS of Rock	Field Tests Laboratory Tests	
	Joint Geometry	Scanline Survey	Joint Dip, Azimuth and frequency
	Virgin Stress *	Measurement	Work carried our previously and assumed to apply mine wide.
	Deformation *	Extensometers	
	Stress Change *	Vibrating Wire, Stress Meters	Instruments installed generally from drill drifts of major stopes
	Condition *	Observation Section sketching	Systematic Visual Inspection of Blast Hole
BLAST GEOMETRY	Design Height	Measured off Plans	Block Dimensions from Layout
	Design Width	Measured off Plans	Block Dimensions from Layout
	Design Length	Measured off Plans	Block Dimensions from Layout
	Dip	Measured off Plans	Block Dimensions from Layout
	Design Tons	Measured off Plans	Block Dimensions from Layout
	Depth Mining	Measured off Plans	Block Dimensions from Layout
	Tons Removed	LDH Bucket Count	Actual Mine Values
	Tons Remaining		Actual Mine Values
	Tons Slough	Estimate	Actual Mine Values
	Back Slough - Type & Amount	Sandplant Records	Generally back slough was observed, while Hanging wall & Footwall
	Hangingwall & Footwall Slough Type & Amount		Slough could only be estimated Based on Tons Overpulled
PRODUCTION DRILLING	Hole Size	Blasting Letters	
	No Full Length	Blasting Letters	
	No Blast	Blasting Letters	
	Burden Spacing	Blasting Letters	
	Total Footage	Blasting Letters	
	Cleaning Footage	Driller Records	Indicator of Potential Ground Problems
	Footage Redrill	Driller Records	
	# Shift Drill	Calculated	
	Avg. Ftg / Shift	Calculated	
	X area @ B' Thru	Measured	
	No Holes B' Thru	Calculated	
	Sq Ft / Hole	Calculated	
	Tons / Ft Drilled	Calculated	Measures of Productivity
PRODUCTION BLASTING	Total Explosive	Measured	
	Lbs Aquamex	Measured	
	Lbs Amex	Measured	
	Lbs Other	Measured	
	No Blasts	Measured	
	No Crater / Slash	Measured	
	Avg. Charge Wt	Measured	
	Avg. Stand off	Calculated	
	Days Blasting	Calculated	
	Powder Factor	Calculated	Measures of Blast Performance
	Avg. Lbs Exp / Blast	Calculated	Measures of Blast Performance
	Avg. Tons / Blast	Calculated	Measures of Blast Performance
PRODUCTION MUCKING	Distance to Orepass	Production Records	
	Size of Scootpass	Production Records	
	Percent Time Operating	Production Records	
	Act. Days Mucking	Production Records	
	# Shifts Mucking	Production Records	
	Tons / Days	Calculated	Productivities
	Tons / Shift	Calculated	Productivities
STOPE DEVELOPMENT	VBM Type		Longitudinal / Transverse
	Length	Drawpoint Data	
	Width	Drawpoint Data	
	HW or FW	Drawpoint Data	
	Height	Drawpoint Data	
	Pillar Width	Drawpoint Data	
EFFICIENCIES	Dilution *	Calculated	Design Tonnage compared with Actual Tonnage
	Recovery *	Calculated	Design Tonnage compared with Actual Tonnage
	Productivity *	Calculated	
	Cost *	Calculated	

NOTE: These parameters recorded seperately and list contained within spreadsheet

TABLE 2: POSSIBLE EFFECTS OF INDUCED STRESS

Magnitude of Induced Stress (MPa)	Possible Effect
25 - 35	Initiation of stress failure in walls of blastholes.
35 - 45	Stress failure in walls of blastholes causing increase in number of redrills and cutoffs. Slabbing of back in drill drifts or walls in drill raise. Slabbing will result in an increase in support requirements.
45 - 55	In wide (hangingwall to footwall) open stopes, average strength of rock mass in final stope of retreat sequence may be exceeded. Heavy support and support maintenance may be necessary to contain possibly fractured ground in back of drill drifts and in drawpoints. Number of redrills may reduce due to drilling "destressed rock". In narrow (hangingwall to footwall) open stopes, the average strength of the rock mass in the final stope of a retreat sequence will probably not be exceeded. Difficult blasthole drilling conditions should therefore be expected. Heavy support and support maintenance may be necessary to contain possibly fractured ground in back of drill drifts in walls of still or raises and at drawpoints. Deteriorating back conditions in cut and fill stopes. Breasting rather than drilling of uppers should be considered. Systematic cable bolting will be required.
55 - 70	In wide (hangingwall to footwall) open stopes rock mass in final stope of retreat sequence probably relaxed, conditions should not deteriorate substantially from those described above. In narrow (hangingwall to footwall) open stopes, average strength of rock mass may be exceeded or be close to peak. Conditions may deteriorate from those described above. Possible severe conditions in back of cut-and-fill stopes. Extensive use of cable bolts envisaged.
> 70	Intact strength of ore exceeded. Slabbing of back of main haulages located with one mining width of stope; of ore passes and fill raises located within one mining thickness of stope; and in crown of draw points.

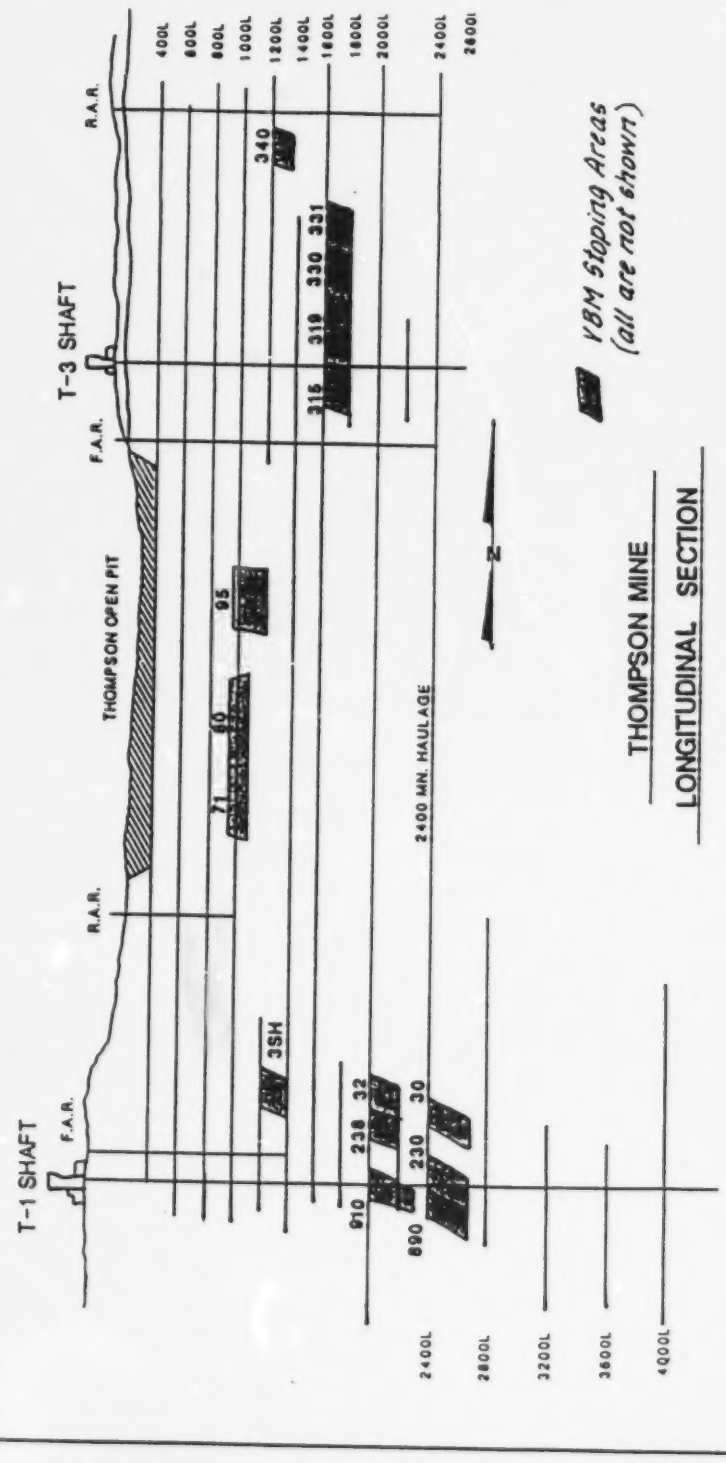
TABLE 3: PERFORMANCE COMPARISON STATISTICS

MINE	ORE BODY CHARACTERISTICS					LAYOUT			PRODUCTION								BREAK DRAW	DILUTION		COMMENTS
	DIP (DEG)	WIDTH (FT)	F'WALL	ORE	STOPE	STOPE HEIGHT (FT)	STOPE WIDTH (FT)	DRAW	DRILL SIZE (INCH)	PATTERN	SPACING BURDEN (FT)	POWDER	DEV (%)	STOPE TONNAGE (TONS)	ADVANCE (FT)	FILL		SMALL ST	LARGE ST	
THOMPSON	70	5-100	SCHIST, WEAK, FOLIATED	SULPHIDE COMPETENT	130	60	15 (LONG RETREAT) UP TO 100	LONGITUD F' WALL	6.5 6.5	FAN	6.5 BLASTHOLE 8 X 10 6.5 BLASTHOLE 5 X 7	1 #/LB/TON	UP TO 3%	20,000	10	CEMENTED 4% TAILINGS 4% CEMENT	NO			CONVERSION FROM CUT - AND - FILL MATURE MINE WITH NUMEROUS WORKPLACES MAJORITY OF STOPING BY LONGITUDINAL RETREAT
HOMESTAKE	70	10-100	SCHIST FOLIATED	SULPHIDE COMPETENT	150	50	15-100	F' WALL	6.5	FAN	5 X 6 TO 10 X 10	1.2LB/TON C COMPETENT 0.6LB/TON IN FRACTURED ORE	3%	100,000	5-10	CEMENTED TAILINGS 6% SLAG	YES	44%	21%	CONVERSION FROM CUT - AND - FILL MATURE MINE WITH NUMEROUS WORKPLACES DOUBLE PATTERN OF PRIMARY AND SECONDARY STOPES
CAROLINEWEST	80 MASSIVE POD	GENERAL COMPETENT	DIORITE JOINTED	140	60	60	WITHIN OREBODY	0.7LB/TON	6.5	FAN	11 X 11	0.7LB/TON	END	70,000	15	CEMENTED TAILINGS 6% SLAG	YES	NO	WALL AND BACK NO STAB A MAJOR PROBLEM	CONVERSION FROM BLASTHOLE EXCESSIVE SLOUGH USING BLASTHOLE OPEN STOPING. MINE LATERAL INSITU STRESSES A PROBLEM
COBAR	75	15-120	CHLORITE BREAKDOWN LIFT TRACE	SULPHIDE COMPETENT	130	200	60	LONGITUD	3.5	FAN	10X 15 1/2	0.5LB/TON FOR SLASHING*	2% UP TO 6%	60,000	30 FT IN SLOT RAISE	CEMENTED TAILINGS 3-6% CEMENT	YES	N/A	10%	CONVERSION FROM CUT - AND - FILL MATURE MINE, SLOT RAISE VCR DRILLED, THEN LONGHOLES

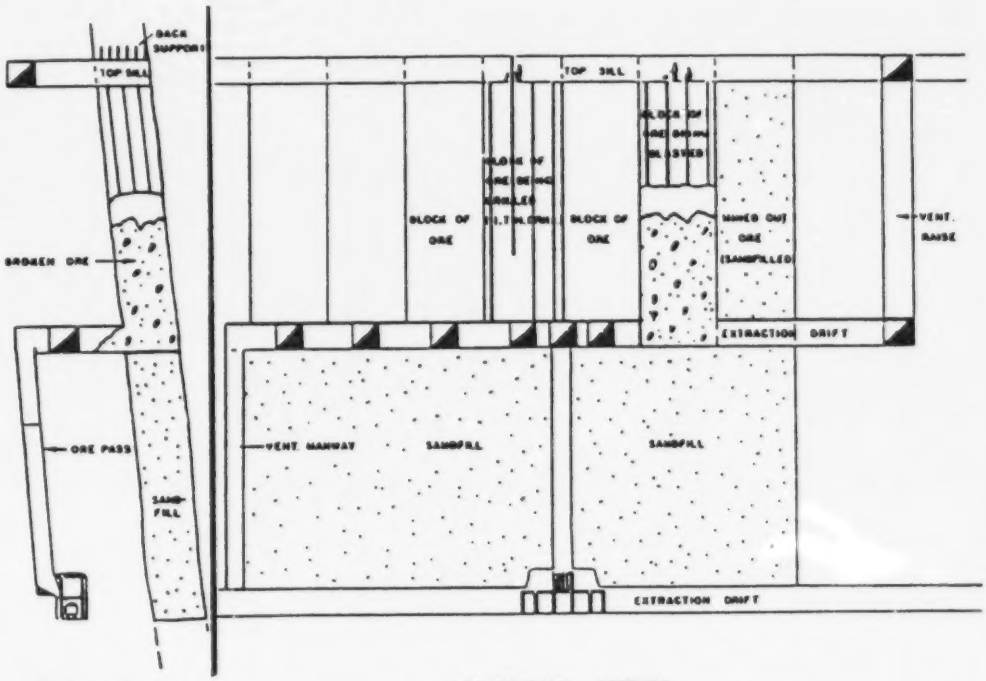
* NOTE RETREAT

THOMPSON MINE
LONGITUDINAL SECTION

Figure 1



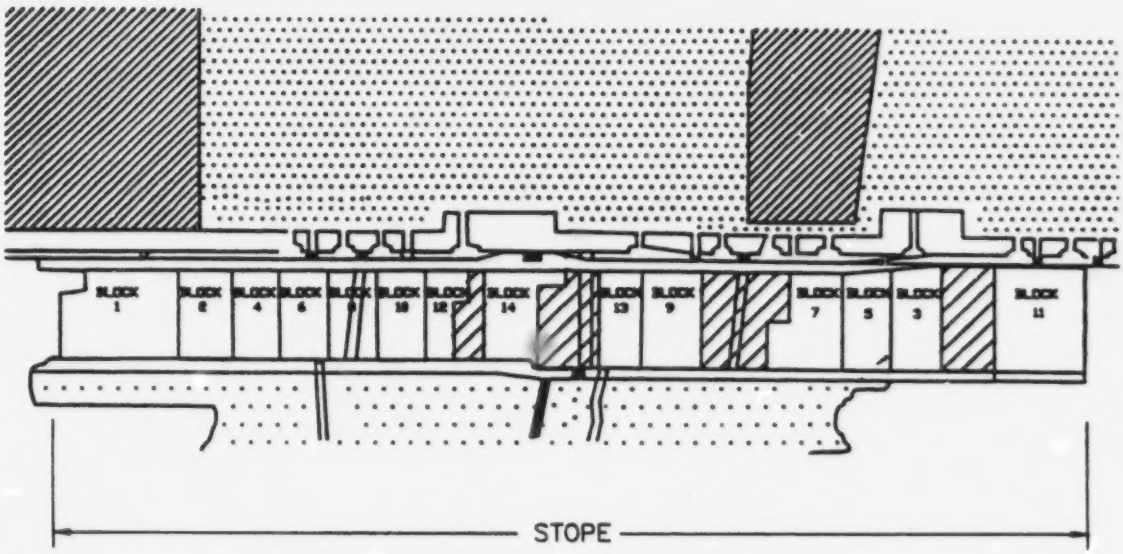
TYPICAL VBM STOPING ARRANGEMENT | Figure **2**



CROSS SECTION

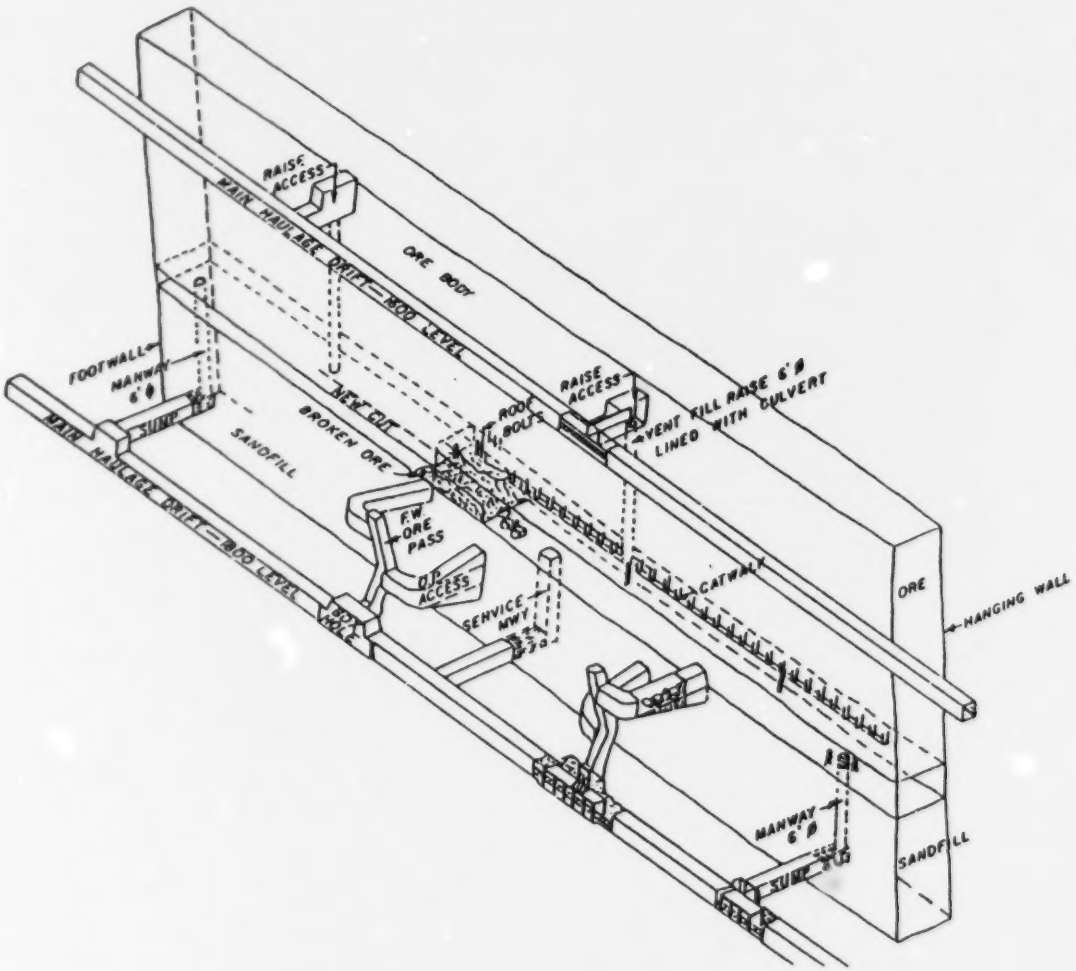
LONGITUDINAL SECTION

VBM BLOCK vs STOPE | Figure **3**

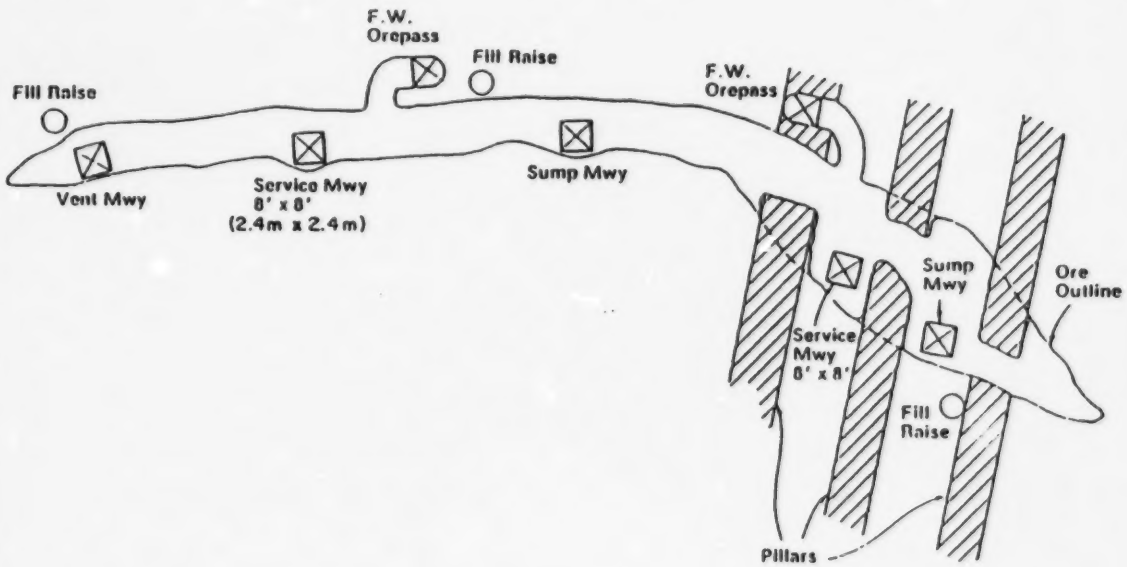


ISOMETRIC VIEW OF A TYPICAL MECHANIZED CUT AND FILL STOPE

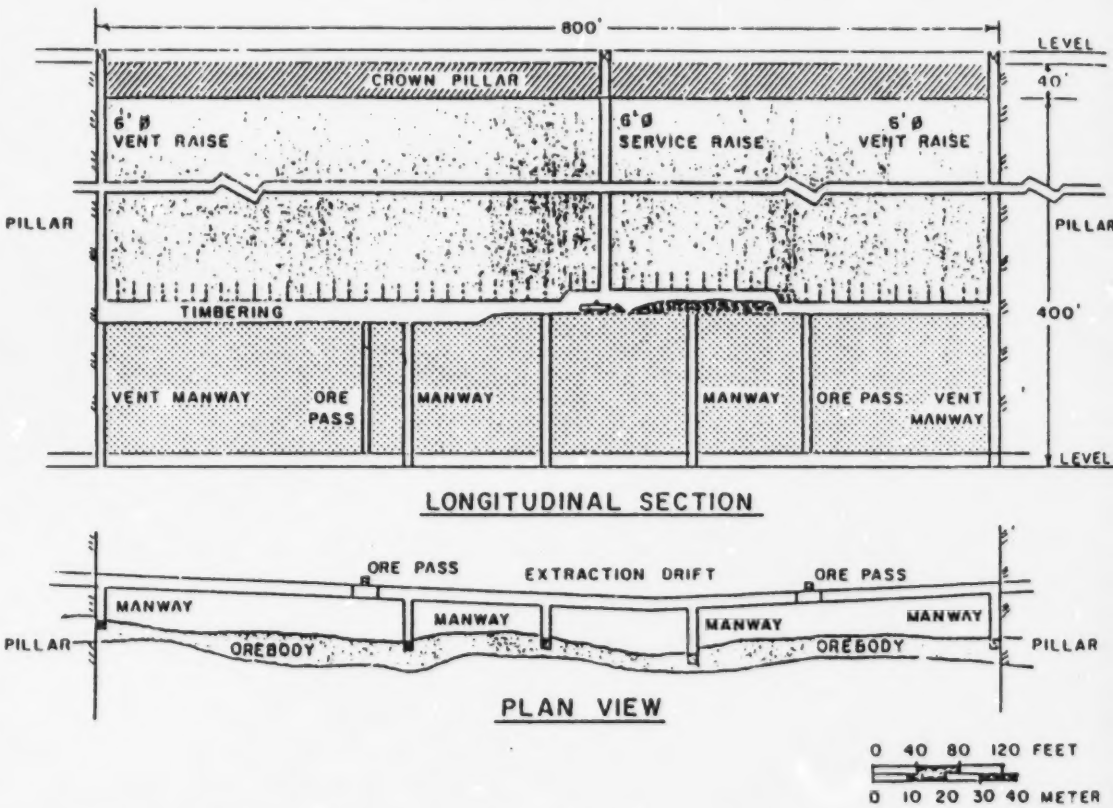
Figure **4**



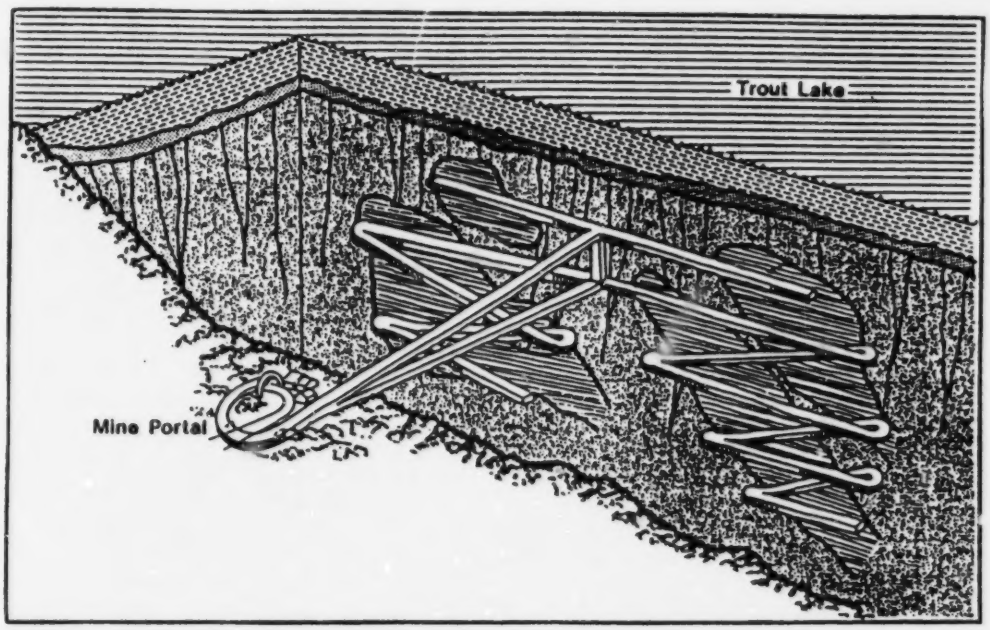
PLAN OF A TYPICAL MECHANIZED CUT AND FILL STOPE | Figure **5**



TYPICAL MECHANIZED CUT AND FILL STOPE | Figure **6**

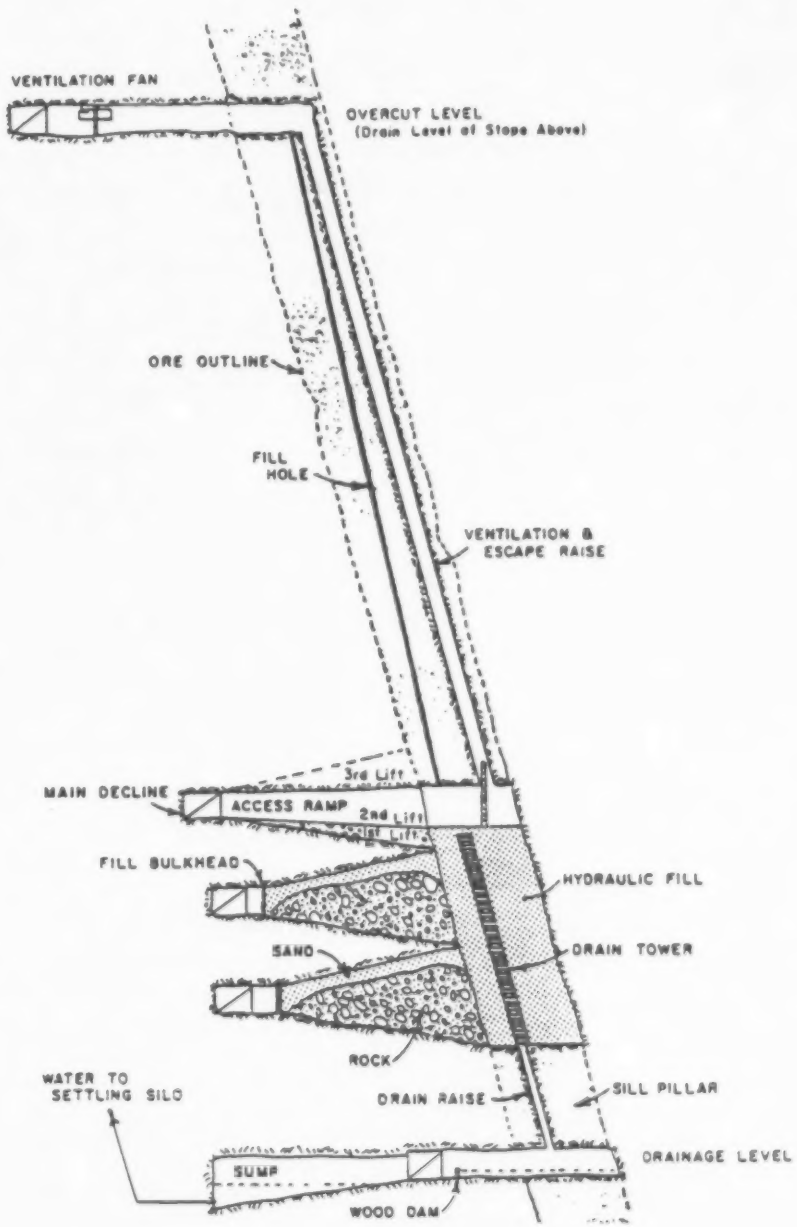


TROUT LAKE OREBODY	Figure 7



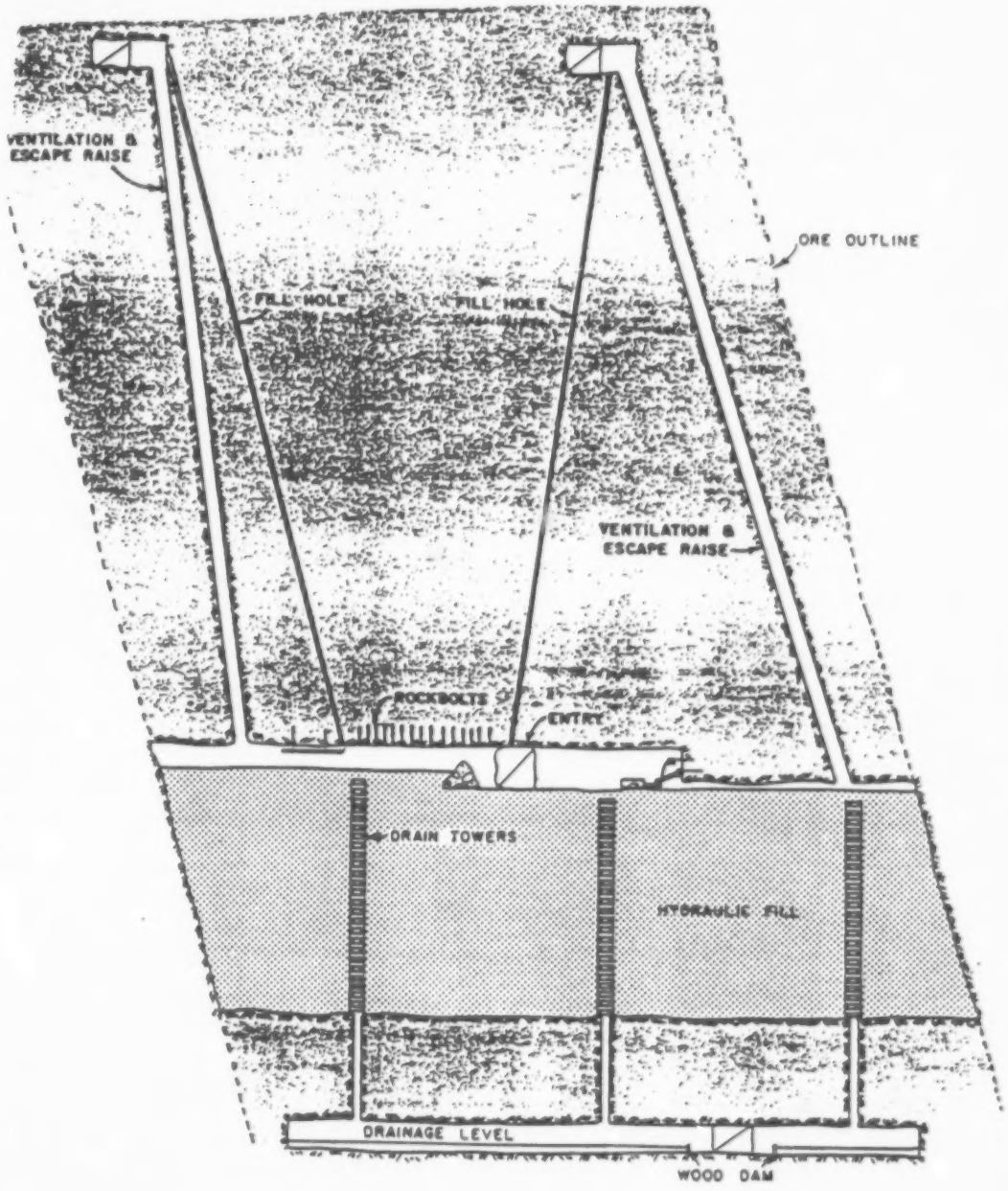
TROUT LAKE MINE
CROSS SECTION

Figure **8**

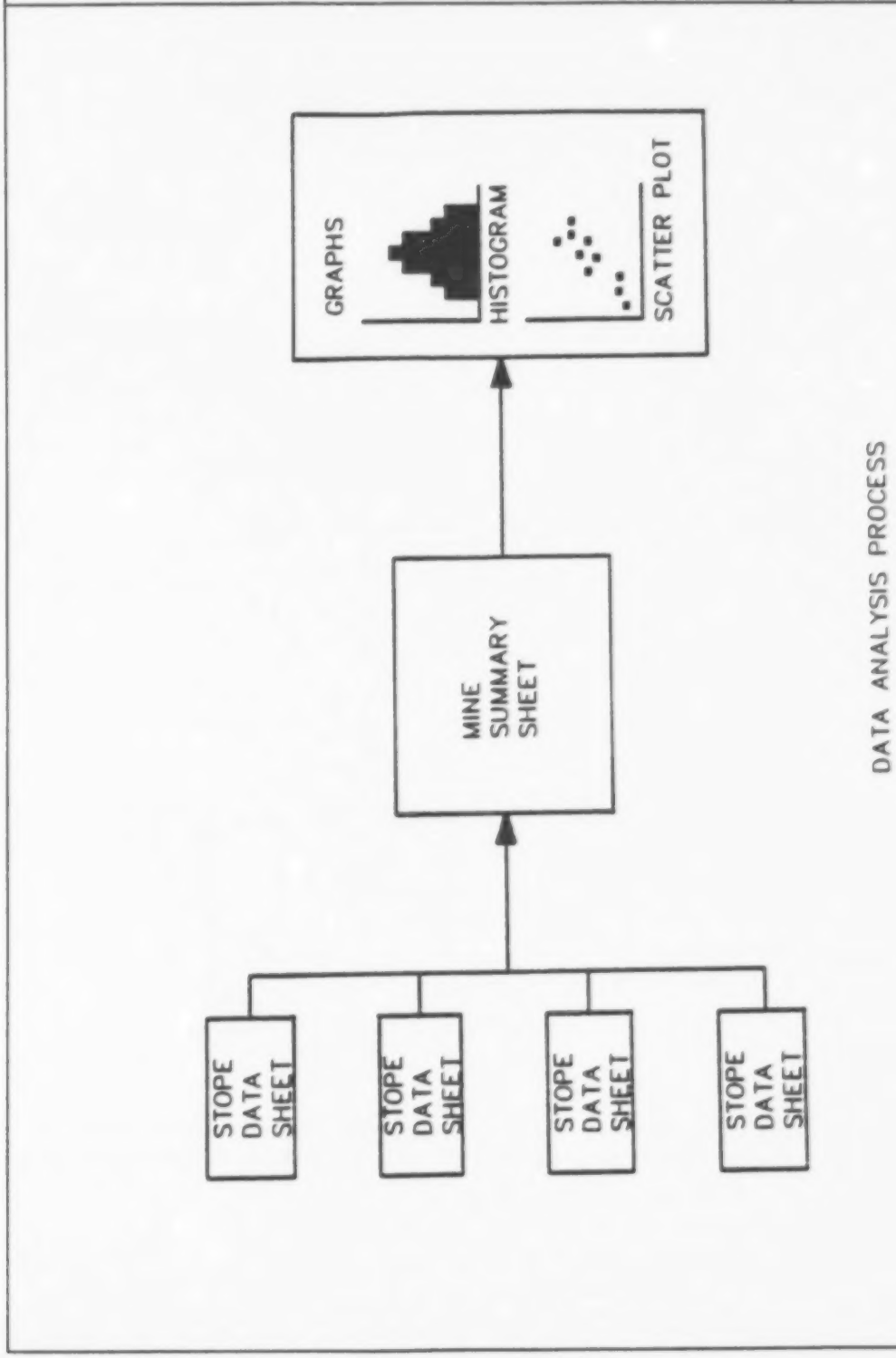


TROUT LAKE MINE
LONGITUDINAL SECTION

Figure 9

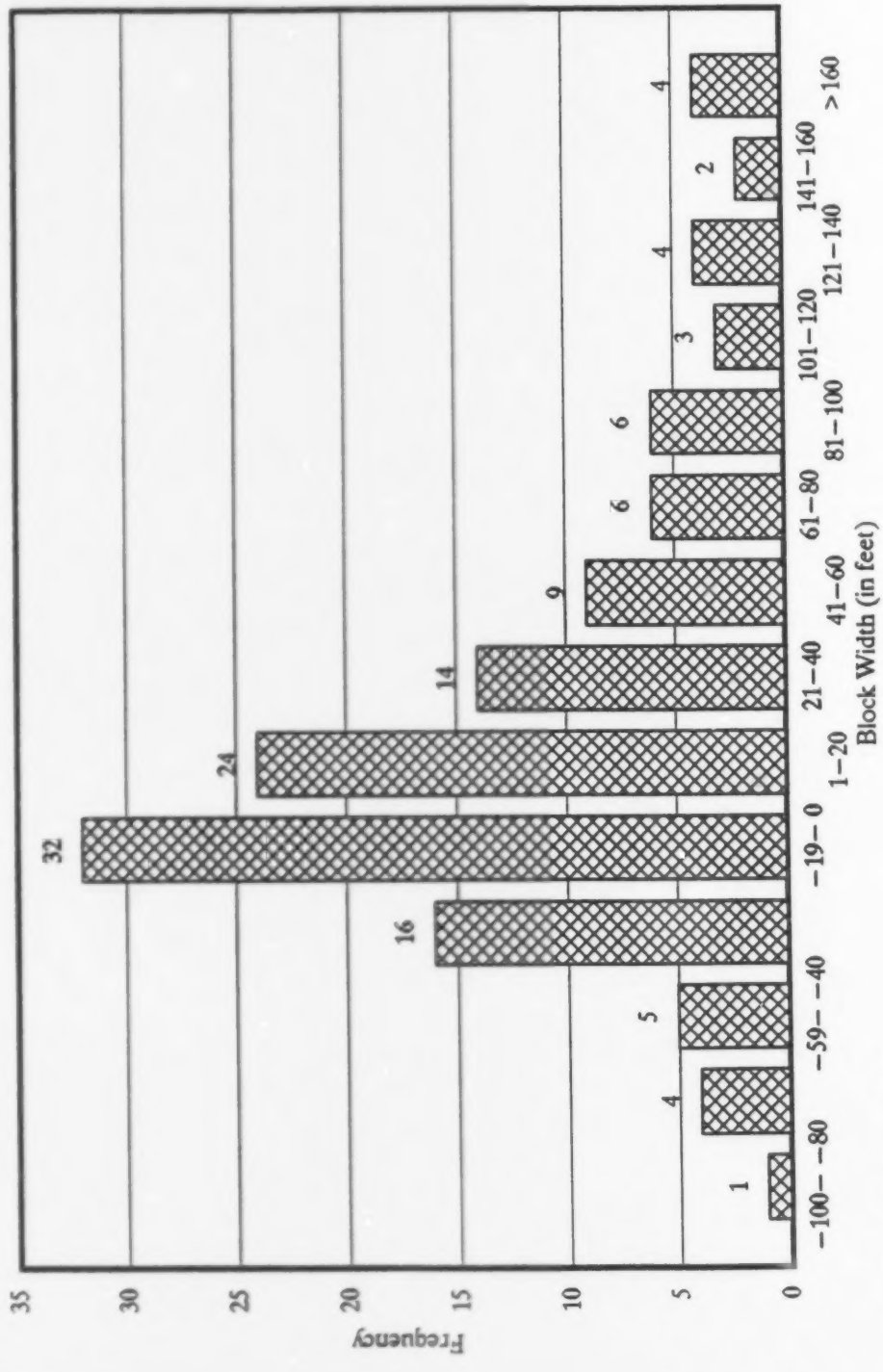


DATA ANALYSIS PROCESS Figure **10**

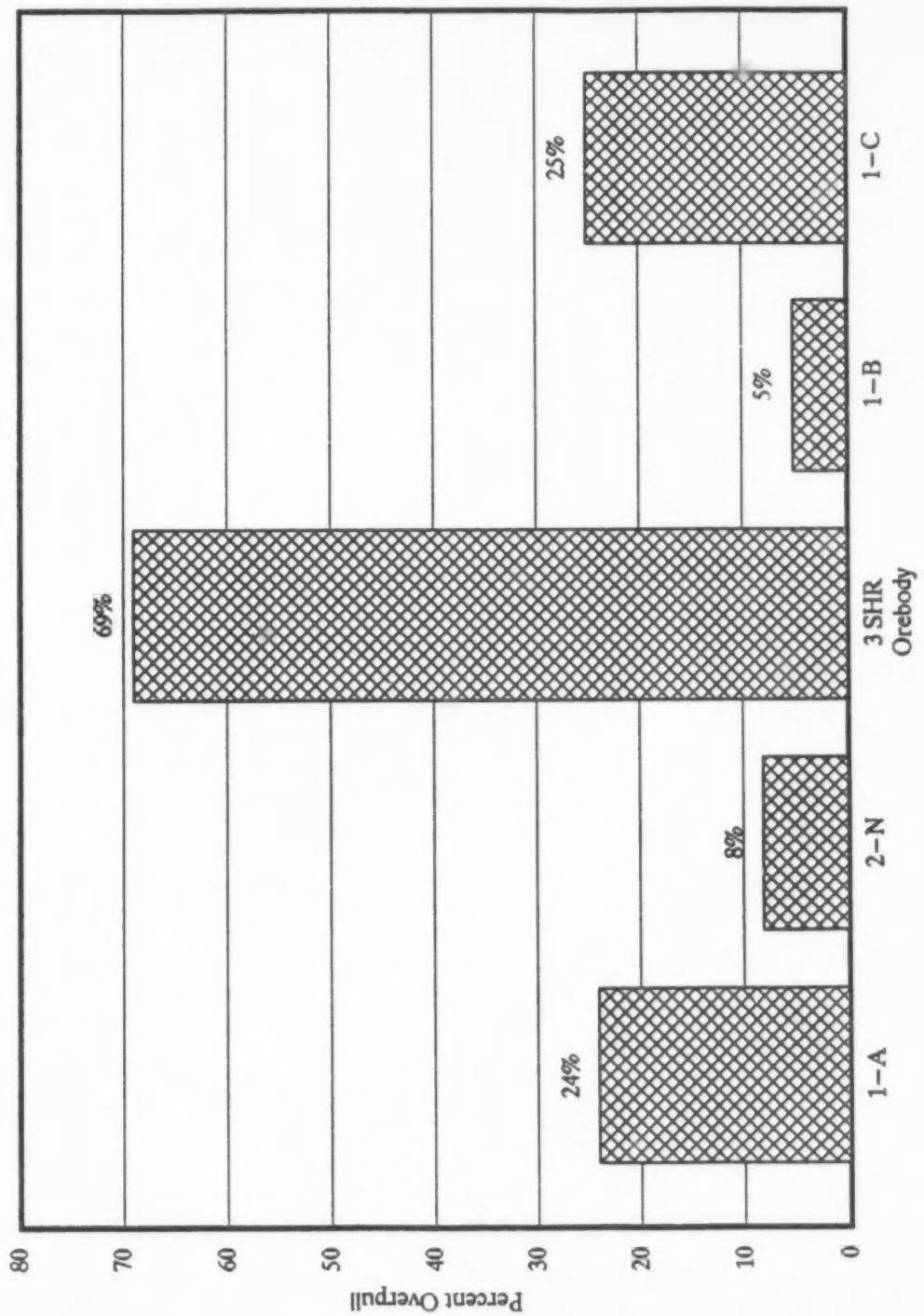


HISTOGRAM PERCENT OVERPULL

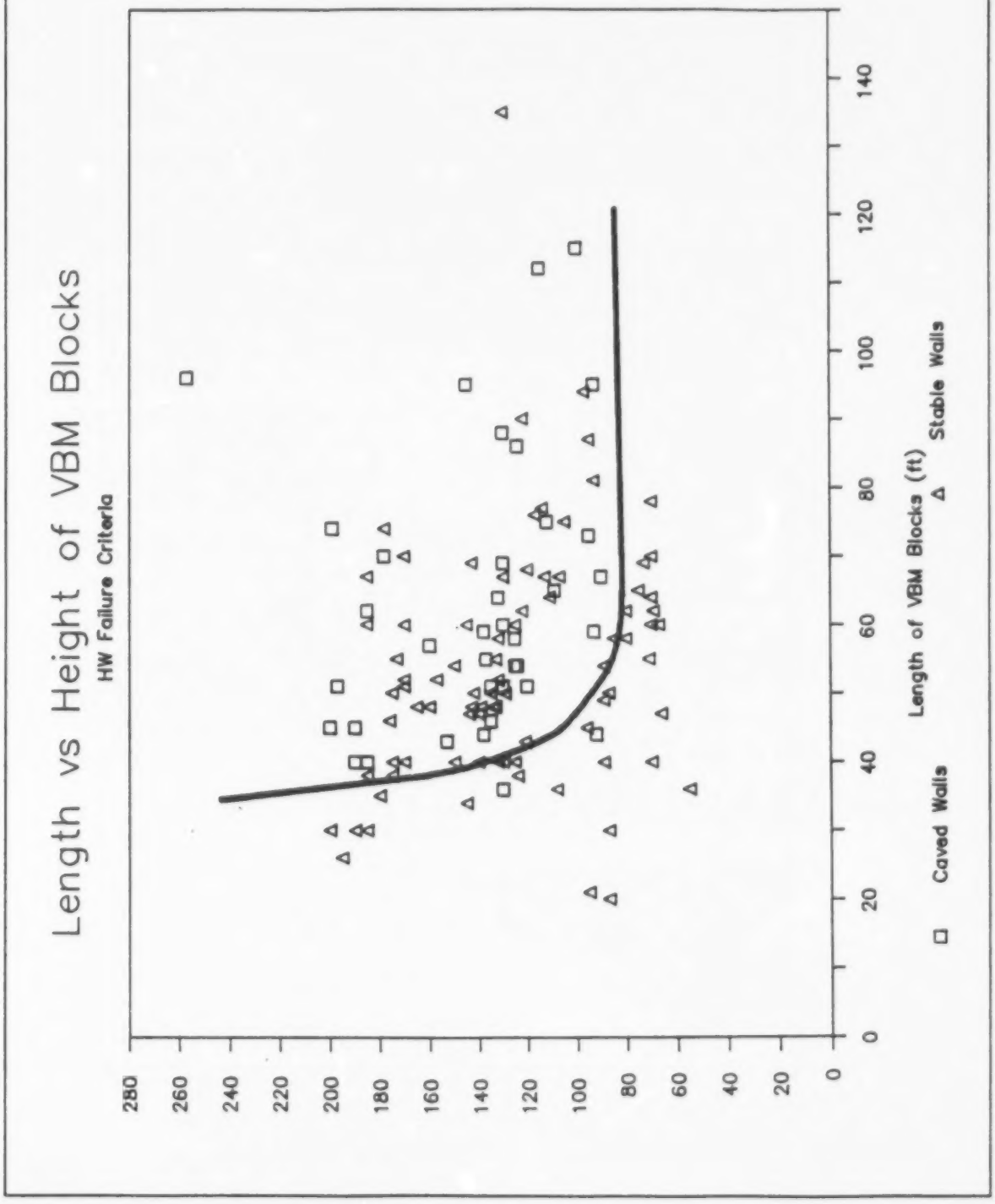
Figure 11



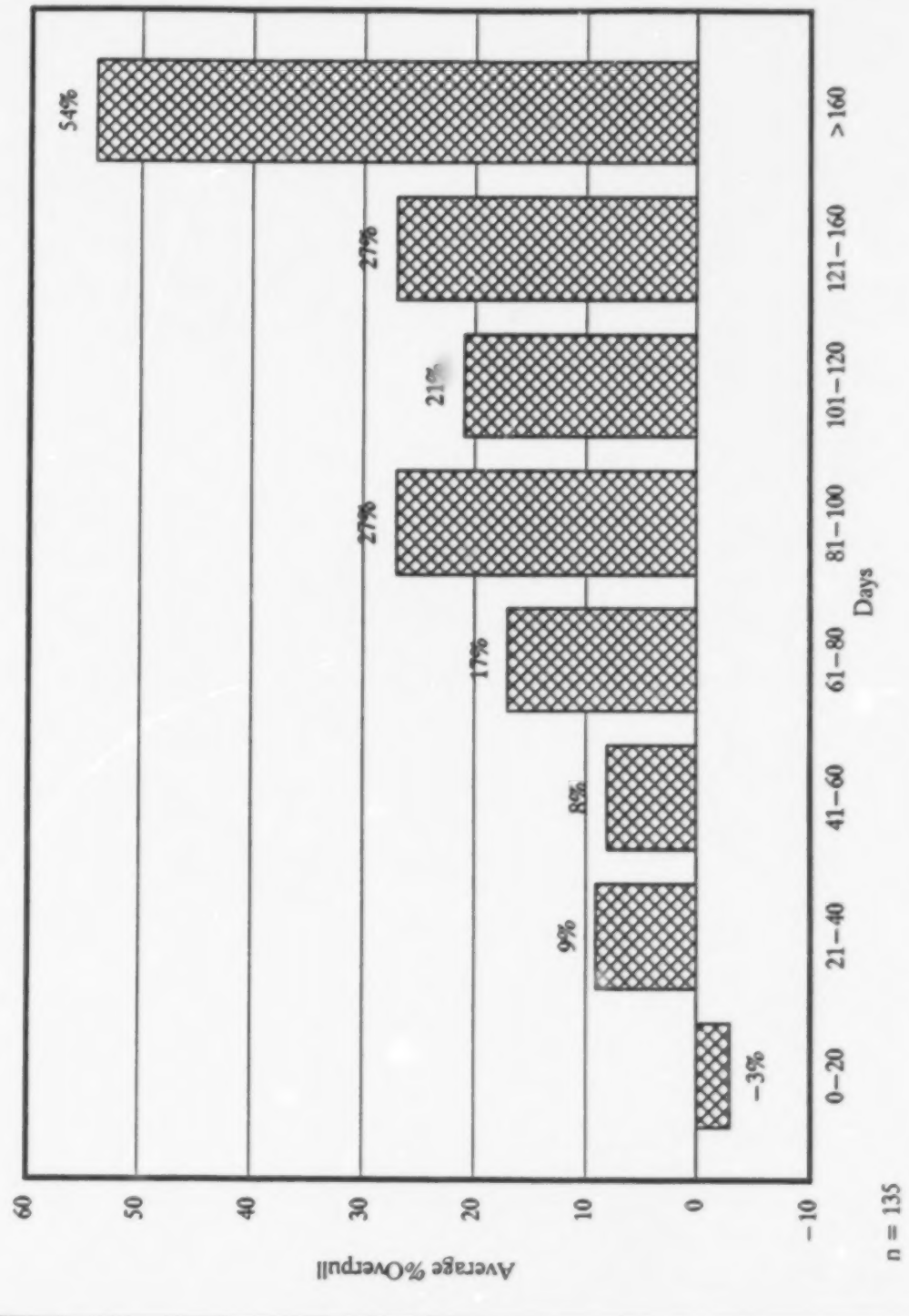
PERCENT OVERPULL BY OREBODY Figure **12**



HANGINGWALL STABILITY vs SIZE AT THOMPSON MINE Figure **13**

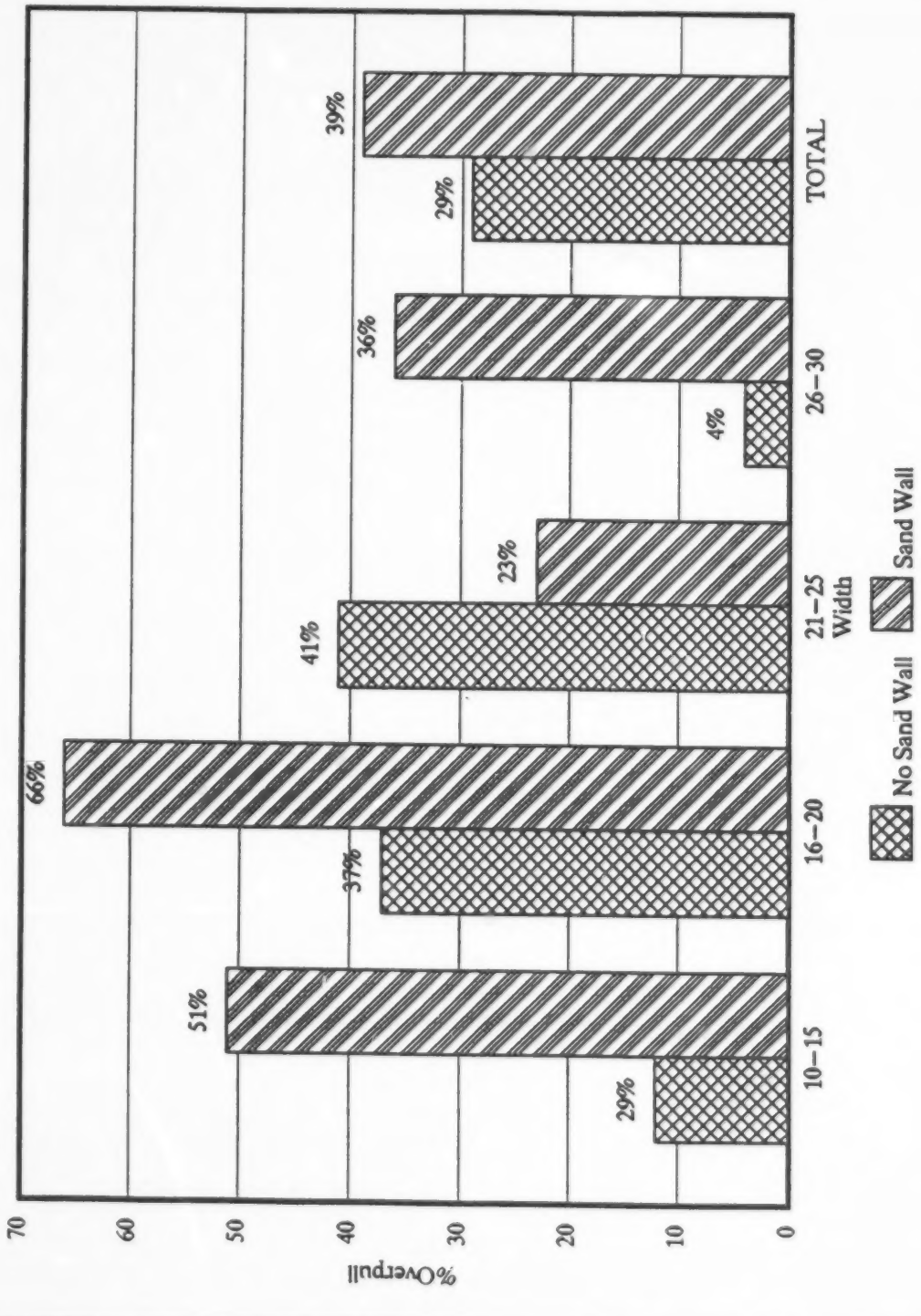


DURATION OPEN (DAYS) vs % OVERPULL Figure **14**



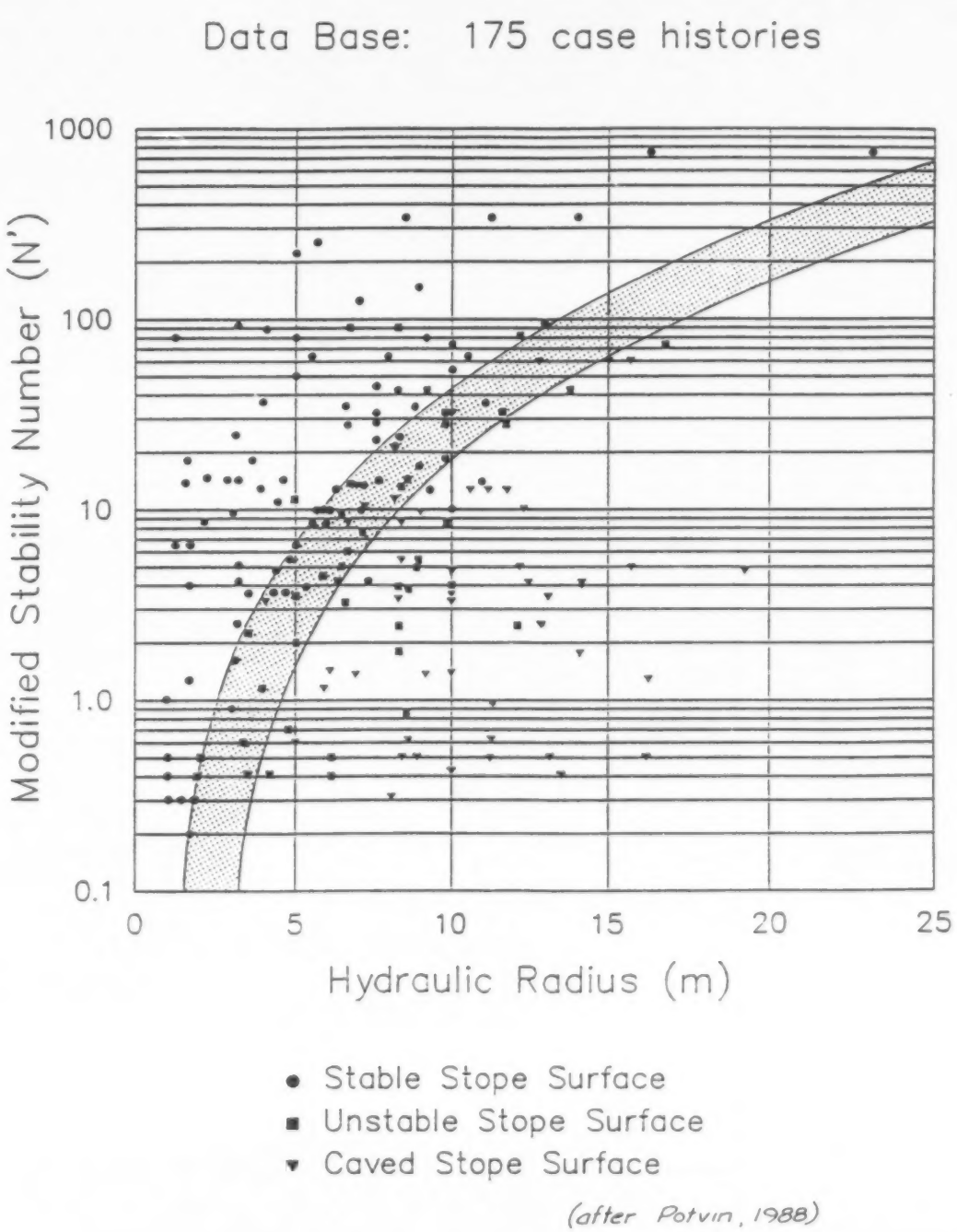
% OVERPULL vs BLOCK WIDTH
NO SAND WALLS vs SAND WALLS

Figure **15**

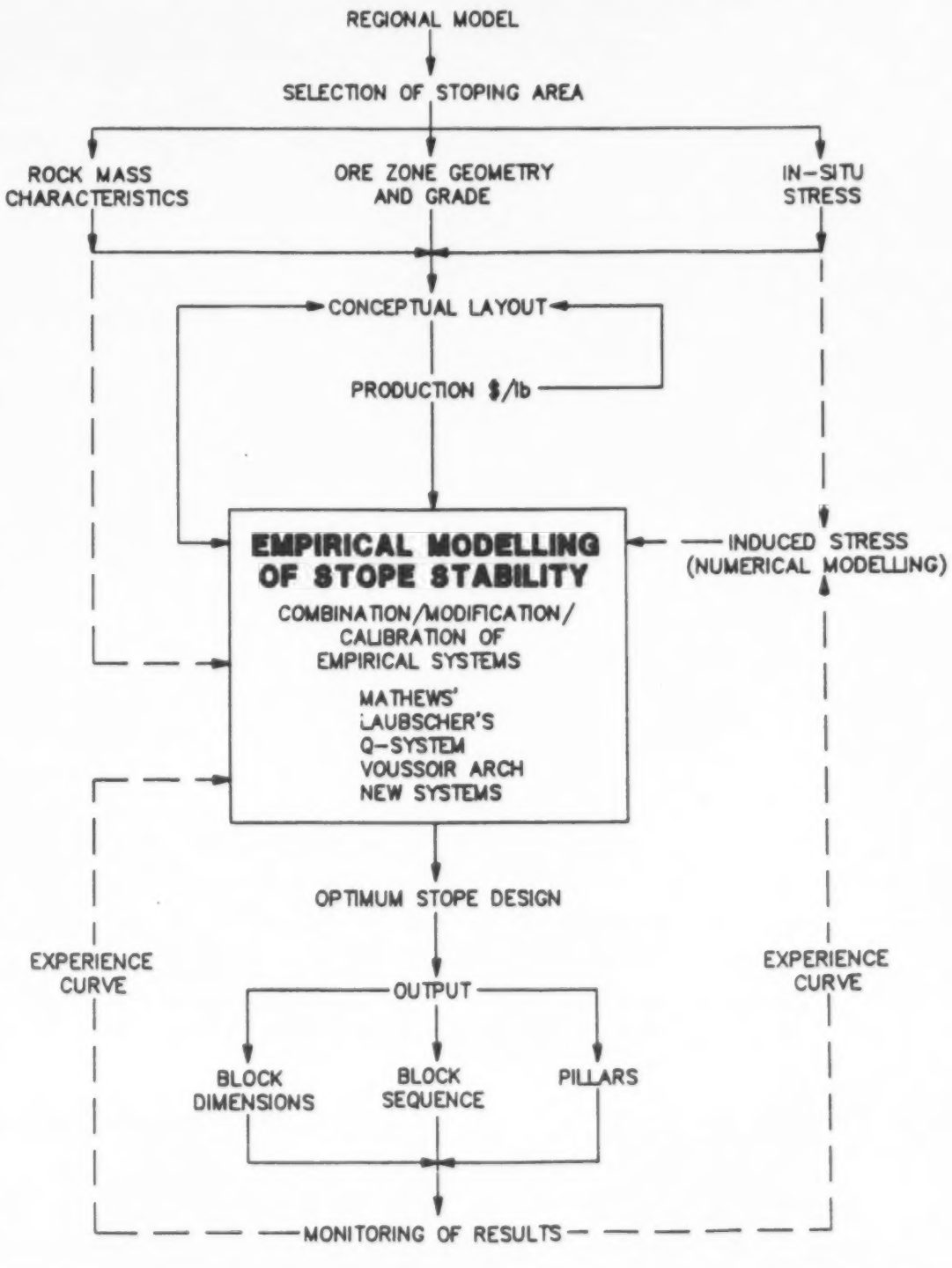


REVISED STABILITY GRAPH FOR MATHEW'S METHOD OF STOPE DESIGN

Figure **16**

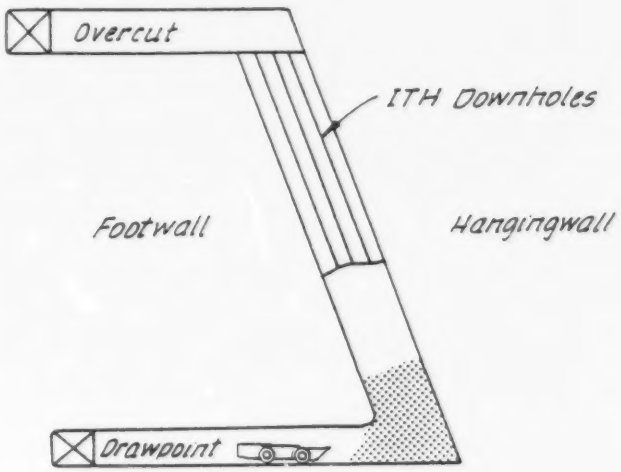


STOPE DESIGN FLOWSHEET Figure **17**

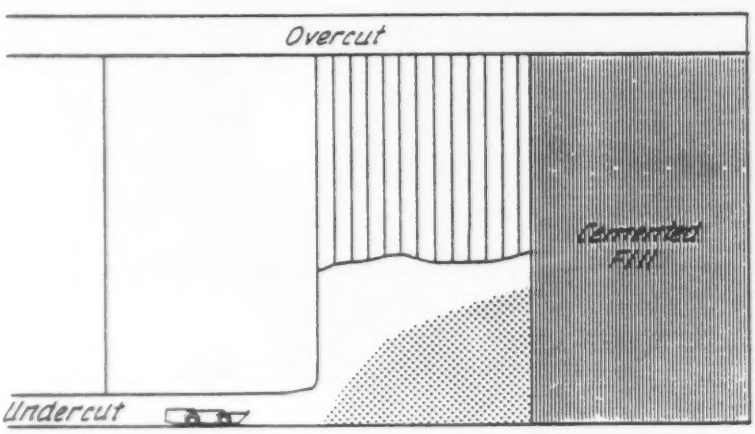


VCR STOPING WITH PARALLEL HOLE DRILLING – THOMPSON MINE

Figure **18**



Cross-Section (Transverse Retreat)

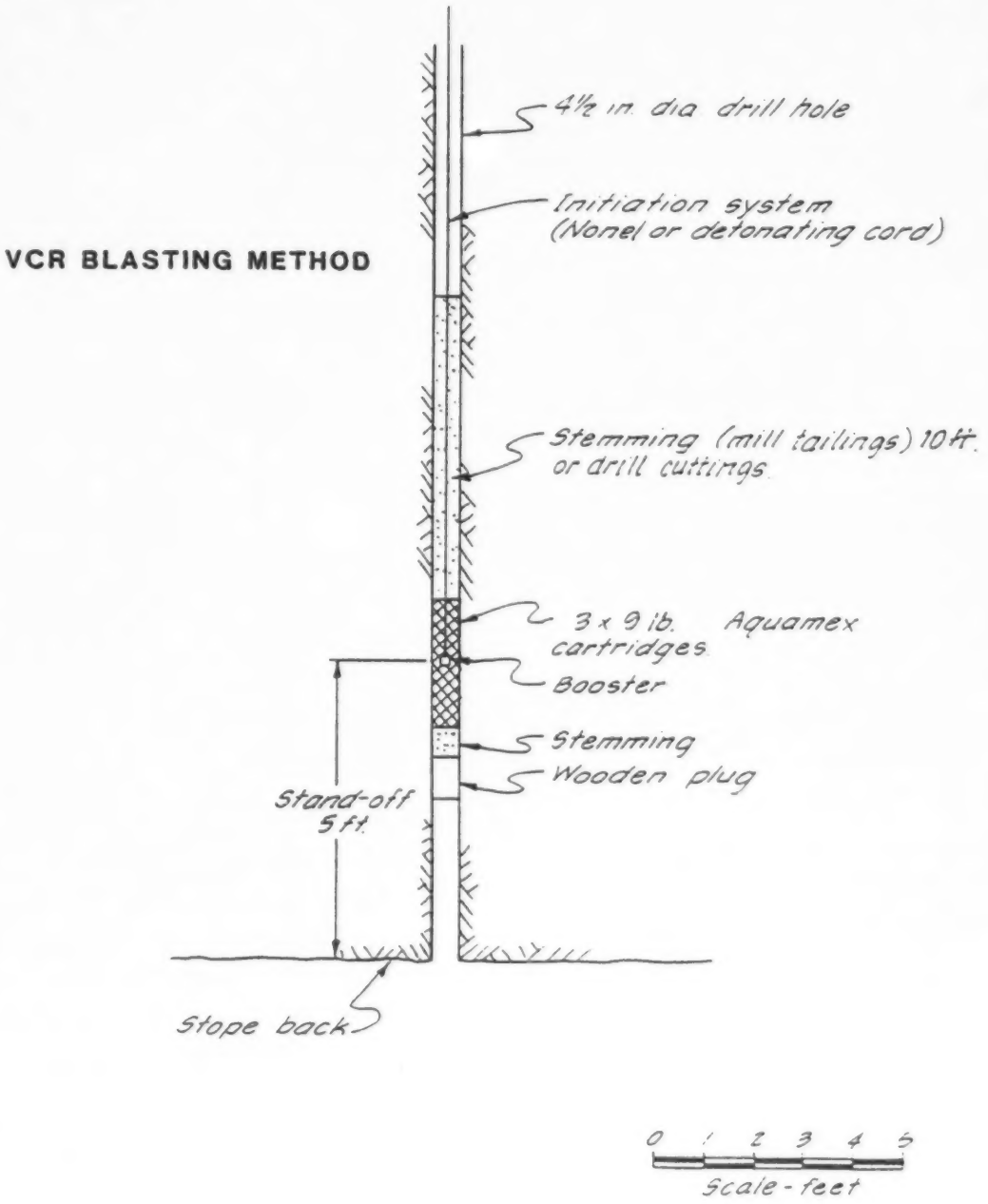


Longitudinal Section (Longitudinal Retreat)

Not to Scale.

CHARGE CONFIGURATION AND GEOMETRY FOR 4.5 inch BLASTHOLE

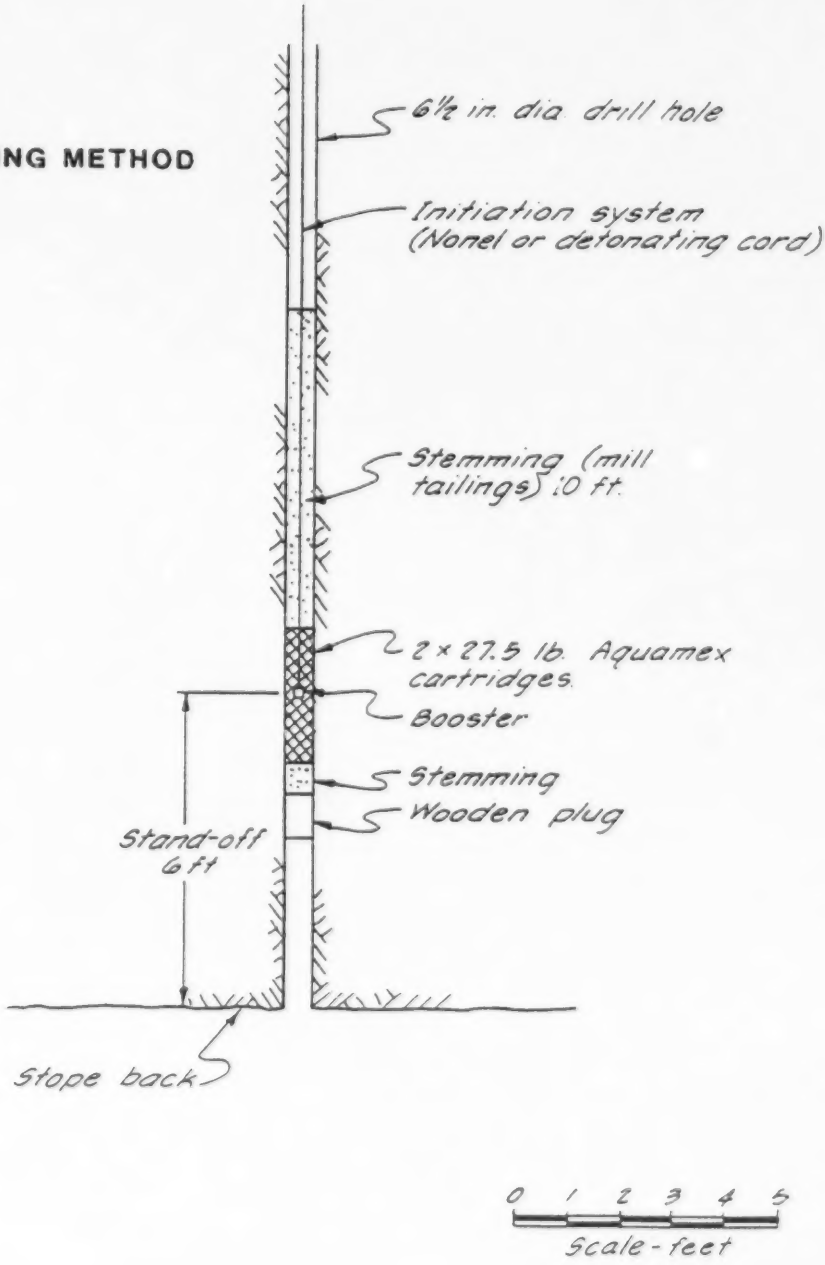
Figure **19**



CHARGE CONFIGURATION AND GEOMETRY FOR 6.5 inch BLASTHOLE

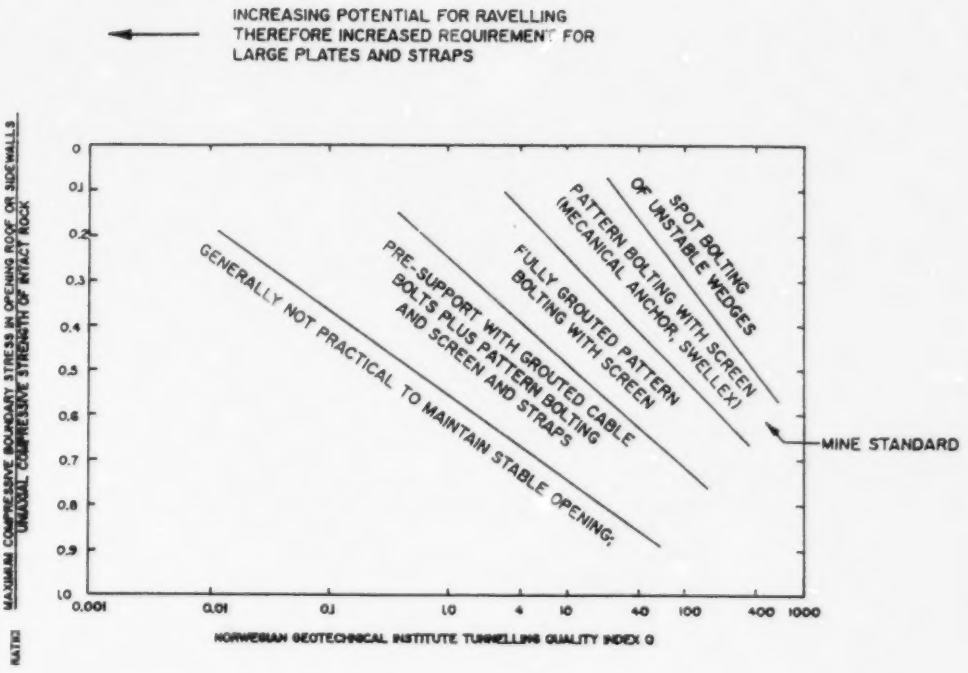
Figure **20**

VCR BLASTING METHOD



EMPIRICAL SUPPORT DESIGN CHART FOR THOMPSON MINE

Figure 21



$$\text{ROCK MASS QUALITY, } Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

1 For excavations up to 4.5 m (15 ft.) span: (barton, 1974)
Minimum bolt length = 2 + 0.05 (span in m) for back.
Minimum bolt length = 2 + 0.05 (height in m) for walls.
Maximum bolts spacing = 1/2 x bolt length.

2 For excavations greater than 4.5 m (15 ft.) span refer to cable bolt design charts.

Reference: Hoek, 1980, and Golder Associates

STANDARD VBM SHOWING STAGES OF LONGITUDINAL MINING

Figure **22**